Decision-Making Models in Mining: The Case Example of the Golgohar 6 Iron Ore Deposit, Iran

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Table of Contents

Figures

Tables

List of Appendixes

Abstract

This research is divided into two general sections. First, the significance and purpose of the study are examined according to conditions of iron ore production and consumption. This part is focused on iron ore resources, extraction and consumption in Iran and the world. Then, comparison of demand and supply of iron ore and reasons why iron ore production will increase are explained, such as population and urbanisation growth, industrial development, new extraction technology, and specially in Iran the availability of inexpensive energy and establishment of new steel plants. Iron ore exploration results in Iran identify new iron ore deposits, such as Golgohar 6, D19 and A13, which have a high amount of overburden. Open pit mining for these deposits is not evident and needs careful evaluation.

The selection of the mining method is very important for mine design, particularly for low-grade and deep deposits. The major factors in the selection of the mining method are the geometry of the ore body, extraction rate, economic and technical factors. A critical assessment of these factors would be helpful for the decision process.

Therefore, the second part of this research provides a systematic evaluation of mining methods in iron ore deposits, with particular focus on the application of fuzzy set theory. Three procedures for the selection of a mining method are applied:

- Qualitative method
- Numerical ranking method
- Decision-making model

This work is mainly focused on a decision-making model based on fuzzy set theory. While the qualitative method mainly defines possible extraction methods, the numeric ranking method and the decision-making model compare alternative mining methods. This approach is applied to the Golgohar iron ore district which is one of four iron ore regions in Iran. In the Golgohar area, six iron ore deposits are recognized by airborne surveying, and Golgohar 6 is one of the deepest deposits. The resource of Golgohar 6 amounts to 65 million tonnes with a grade of 56% Fe. The thickness of the orebody is about 100 m and the depth is between 500 to 600 m below surface. Exploration

activities for Golgohar 6 include geological investigation, geophysical measurements (airborne surveying, magnetometry and gravimetry), core drilling, sampling and analysis.

The application of the above selection procedures for the Golgohar 6 deposit defines that both the surface mining and the sublevel stoping method are appropriate extraction methods. The preliminary economic study of surface mining with shovel and truck versus surface mining with semi-mobile crusher and conveyor as well as sublevel stoping constrains the most suitable mining method for the current information level of the Golgohar 6 exploration project. However, the density of the current exploration data (indicated mineral resources) is not sufficient for making a final decision on the mining method. The continuation of exploration activities and more exploration drilling (the spacing between the drill holes should be 50 m or less) are necessary.

1 Introduction

The ever-increasing demand for iron ore has increased both its exploitation and price. The higher price for iron ore is an impetus for engineers to look for reserves located at greater depths. For such reserves, the selection of the mining method is of great importance, and it results from the consideration of different factors, including the size of the overburden and reserve, the dip of the orebody, the deposit shape and rock mechanical properties of the ore horizon and the surrounding country rock.

Considering the current trends of economic globalization, the survival and growth of the mining sector depends heavily on creating competition among such activities. This, in turn, entails access to technological capacities, the training of human resources, developing of the infrastructure and financial sources and the diversity of products.

The existence of multitudes of energy resources such as oil and gas, access to educated labour, existence of ore reserves and easy access to shipping transportation through the ports in the south of Iran have provided Iran with advantages for developing its mining industry.

The exploitation of deep iron ore reserves in Iran appears extremely important due to the following reasons:

- Increase in the demand for iron ore in recent years
  Due to the increase in steel consumption in Iran, this issue is especially significant. Iranian steel consumption in 1993 was 3 million tonnes and increased to 18 million tonnes in 2013. This increase was due to reasons that include the growth of the construction, automobile, and transportation industries as well as the increase of plants and industrialised equipment. Since Iran is considered a developing country, it is predicted that the increase in demand will continue in the future. The production and consumption of steel are forecast to increase worldwide.
- The increase in the price of iron and iron ore in recent years
  In general, all types of raw materials have experienced a price increase in recent years. However, a comparison with the inflation rate illustrates the significant increase in the steel price: the annual average increase of the steel price

during last ten years was 18% and in the same period, the world inflation rate was estimated to be 4%.

- Construction of several new steel-producing factories
  Considering the needs of the steel industry, many plants, including coal coke production facilities, pellet production plants, iron ore processing facilities and steel production plants have been established or are under establishment in Iran. Accordingly, the demand for iron ore will rise. According to the Iranian Mines and Mining Industries Development and Renovation Organization (IMIDRO), the demand for iron ore in Iran will be 70 million tonnes in 2020.
- Existing deep iron ore reserves such as iron ore anomaly Golgohar 6
  Given the development and growth of mining and the related technology, the conducting of technical studies to investigate the possibilities of extracting deep resources of iron ore seems necessary. The extraction of such reserves could ultimately be possible through the consideration of technical as well as economic parameters.
- The population and urbanization of the world will increase in the next 40 years. On the other hand the population of Iran has doubled in the last 40 years and will grow in the future. In 2050, world population will be greater than 9 billion. Therefore steel consumption will increase along with population and urbanization growth.

Based on abovementioned reasons, investigation of new iron ore deposits and determination of the mining method is necessary and a suitable pattern would be helpful for selection of the best mining method. The purpose of research is focused on preparation of a suitable pattern for investigation of iron ore deposits based on ore demand and economical situation and deposit parameters. The framework of research is shown in Figure 1 as sequence investigation diagram. For this purpose, at first the reasons for an increase in ore production must be considered. Then ore deposit information (options) such as exploration data and ore body geometry, infrastructure and parameters will be collected. On the other hand, when an exploration programme is carried out to get to the next stage, the value of a property may be enhanced, reduced, or remain the same, depending on how the results of the programme affect the perceived exploration potential. For this assessment, the extraction method used must be determined. Needless to say, feasibility studies need to be carried out before any step and immediately after selecting the most suitable extraction method (open pit mining versus conventional underground mining).

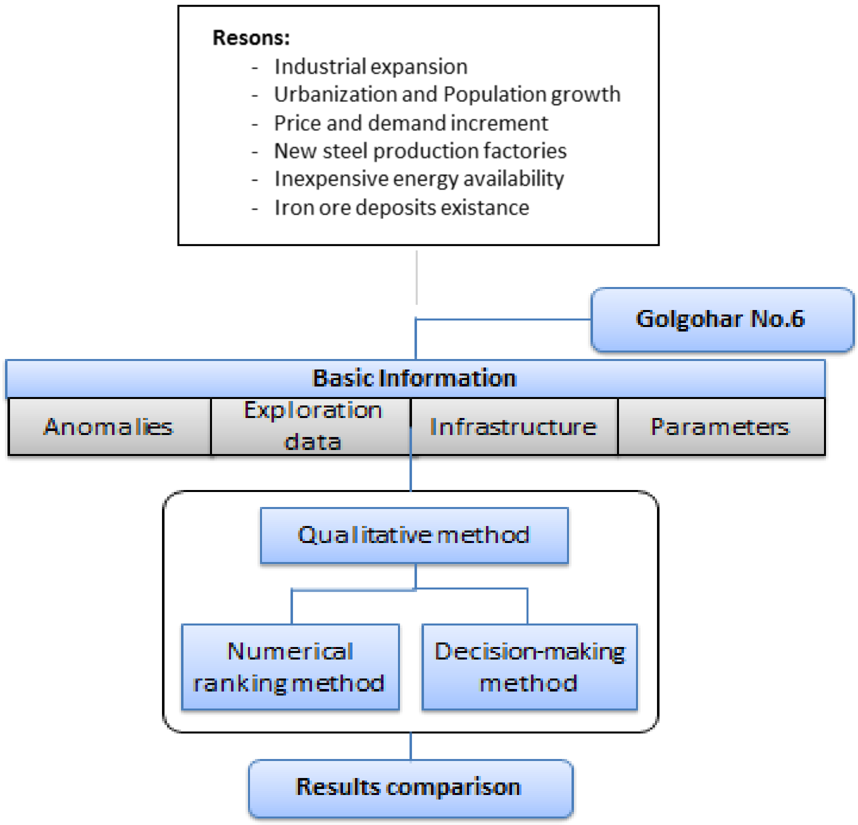


Figure 1 Investigation sequence of research framework

The extraction methods are assessed: Qualitative method, Numerical ranking method and Decision-making method. Appropriate methods are first determined qualitatively. Then, appropriate methods will be determined by the numerical ranking and decision-making method. In recent years, the Fuzzy Set Theory has been widely used in order to make such decisions and choose the most suitable method. Although the Fuzzy Set Theory has been applied in many projects, it is not still common as a final tool for selecting the extraction method in the mining sector. This research is focused on usage of the decision-making method in mining method selection. Several parameters, such as shape, depth, size, rock mechanical properties, environmental issues and recovery factor, and possible exploitation methods are introduced as criteria and alternatives, respectively. The next step of the sequence is the comparison of the results of the numerical ranking and decision-making methods.

In this research, Fuzzy Set Theory is introduced and then applied for the selection of the most suitable exploitation method for the anomaly Golgohar 6. With a longitude of 55° 20` and latitude of 29° 05`, Golgohar 6 is located in Kerman Province, Iran. The exploration activities conducted in this zone consist of geological and geophysical surveys, core drilling, sample acquisition as well as laboratory tests and analysis.

2 Iron Ore Situation

Given the increasing use of iron ore and the rise of global steel consumption, the iron ore supply cannot satisfy market demand. On the other hand, considering the world competition in providing a more inexpensive production, the production of ore with lower grade or the extraction of deeper anomalies have increased in importance.

Due to the position of steel in the economy and industry as a strategic product as well as being the factor influencing the indicators of the development of countries, different investments have been made in this type of industry. The production of steel has drastically changed during the past 50 years. Although iron is the one of the most ancient metals used by human beings, new extraction and quality improvement methods for iron ore are still under study.

Iron is one of the most abundant rock-forming elements, constituting about 5% of the Earth's crust. It is the most abundant and widely distributed metal and is indispensable to modern civilization, and people have been skilled in its use for more than 3,000 years. However, its use only became widespread in the 14th century, when smelting furnaces (the forerunner of blast furnaces) began to replace forges.

Iron is one of the three naturally magnetic elements; the others are cobalt and nickel. Iron is the most magnetic of the three. The mineral magnetite (Fe_3O_4) is a naturally-occurring metallic mineral that is occasionally found in sufficient quantities to be an ore of iron.

The iron ore mines found in various locations extract iron ore deposits in forms that include hematite ore, taconite ore, goethite, magnetite, etc. The iron occurs as oxides, carbonates, sulphides and silicates. Commercially, oxides are more important. The main minerals that contain iron are Magnetite(Fe_3O_4) with 72.4% iron content, Hematite (Fe_2O_3) with 69.9% iron content, Siderite ($FeCO_3$) with 42.8% iron content, Limonite and Goethite ($Fe_2O_3.H_2O$)with 62.9% iron content and Pyrite (FeS_2) with 46.5% iron content.

Iron ore should have a high grade to be used as a source of iron. For example, nowadays, reserves with less than 25% Fe are not extracted. However, the volume of the reserve as well as other parameters such as ease of extraction can highly affect the extractability of an ore deposit.

About 98% of iron ore is used to make steel - one of the greatest inventions and most useful materials ever created and the consumption of other usage of iron and iron ore are very low. Powdered iron: used in metallurgy products, magnets, high-frequency cores, auto parts, catalysts. Radioactive iron (iron 59): in medicine, tracer element in biochemical and metallurgical research. Iron blue: in paints, printing inks, plastics, cosmetics (eye shadow), artist colours, laundry blue, paper dyeing, fertilizer ingredient, baked enamel finishes for autos and appliances, industrial finishes. Black iron oxide: as pigment, in polishing compounds, metallurgy, medicine, magnetic inks, paints, in ferrites for the electronics industry.

The value of an iron ore deposit depends on four factors: grade, extraction terms, processing possibility and transport to the market. Metallurgically, the extracted iron ore should physically and chemically satisfy the needs of steel production industries. Grain size, physical properties, mechanical strength, presence or absence of detrimental elements (phosphorus, sulphur or some metals, such as lead and zinc) and waste can determine the quality and operational conditions of the smelting system.

The iron ore used in steel industry can usually be found in two forms:

1- Lump ore, which is usually prepared dry using a crusher and magnetic separator. For the purpose of this operation, usually iron ore with a relatively high grade (above 50% Fe) and lower levels of detrimental elements, such as phosphorus and sulphur is used. Lump ore is used in blast furnace systems in which the direct reduction method cannot be applied.

2- Pellets, which are produced by the agglomeration of fine minerals and concentrated iron mineral with normal size of 10 to 50 mm. The grade of pellets is about 64% Fe, higher than that of crude iron ore. The mechanical strength of pellets should be so high that it prevents them from being grinded on loading in the blast furnace. Pellets are used in steel production for two main reasons: reduction of energy consumption and reduction of slag.

Therefore, all sources of iron used by human industry exploit iron oxide minerals; the primary form used in industry is hematite. Based on magnetic property of magnetite, the magnetic separation is easy and normal method for separation of ore and gangue minerals.. Due to the high density of hematite relative to silicates, beneficiation usually involves a combination of crushing and milling as well as heavy liquid separation. This is achieved by passing the finely crushed ore over a bath of solution containing bentonite or other agents that increases the density of the solution. When the density of the solution is properly calibrated, the hematite will sink and the silicate mineral fragments will float and can be removed.

Four main type of iron ore deposits are extracting, magnetite, titanomagnetite, massive hematite and pisolitic iron stone deposits, and the extraction method of iron ore generally related on the type of deposits. /1/.

The manner in which iron ore is extracted from mines depends on the type of ore deposits in the mine. The amount of hematite needed in any deposit to make it profitable to mine must be in the tens of millions of tonnes. Limonite, siderite and other iron minerals have little application because of their low iron grade. Taconite is a silica-rich iron ore that is a low-grade magnetite ore. However, the iron-rich components of such deposits can be processed to produce a concentrate that is about 65% iron, which means that some of the most important iron ore deposits around the world were derived from taconite. Taconite is mined in the United States, Canada and China.

The annual global volume of iron ore production is 3 billion tonnes and is increasing as the world and urban populations grow. The volume of iron ore production has been increasing during the past decades and in 2005 the volume of production was 1,500 million tonnes, which is currently doubled.

Today, iron ore is extracted in 50 countries and mostly 3/4 of the total iron ore is extracted in seven countries, among which China is in the first place with an annual production of 1,300 million tonnes. The global volume of iron ore reserves and iron content are estimated at 170 billion tonnes and 81 billion tonnes, respectively. Australia has the largest iron ore reserves in the world, at 35 billion tonnes.

Iran has 2,500 million tonnes of iron ore reserves and produces 38 million tonnes annually. The steel industry in Iran was established in 1930 with the help of German corporations. However, it ceased operations as World War II started. In 1971, steel production began in Isfahan when the factory was equipped with an electric air furnace to increase the volume of production to 600,000 tonnes. The project was able to achieve a volume of 1.9 million tonnes. Along with steel production, iron ore quarries in Chadormalu and Choghart were discovered in Yazd province. After that, with establishment of Khuzestan Steel, Mobarakeh Steel and other plants, the annual production increased to 15 million tonnes.

2.1 World Iron Ore Status

The world iron ore market will be characterized by tight conditions and in the next few years, there will be a gradual adaptation of supply to a continuously growing demand by the addition of new capacity.

The demand shock that has precipitated the increase in the price of iron ore has been due to growth in China's construction, investment and manufacturing sectors, and this growth is likely to continue for some time.

Steel production growth has stimulated demand for the key inputs into steel production: This has resulted in dramatic price increases about 70% for iron ore compared to before 2005. In addition, global prices for thermal coal grew in recent years in response to power companies, especially in Asia, competing to secure additional coal feedstock to meet the growing electricity demand /3/.

The cost to produce steel varies from country to country, largely with the cost of raw materials, as well as labour and energy. Russia, India, Ukraine, Brazil and Iran are able to produce steel at the lowest cost because of a combination of cheap energy and labour. Unlike many other commodities, China is not the lowest-cost steel producer. Chinese labour is cheap, but energy in China is still pricey. As a result, China produces steel at a cost 20% below that in the US, but energy costs in China are higher than they are in Russia, India, Ukraine, Brazil and Iran. The top ten steel producers and consumers are shown in Table 1 .

Table 1 The major steel producer and consumer countries in 2013 /1/

Production		
	Country	(Mt)
1	China	779
2	Japan	110
3	USA	87
4	India	81
5	Russia	68
6	South Korea	66
7	Germany	43
8	Turkey	35
9	Brazil	34
10	Ukraine	33
11	Italy	24
12	Taiwan	22
13	Mexico	18
14	France	16
15	Iran	15

Import		
	Country	(Mt)
1	USA	18
2	Thailand	14
3	Indonesia	12
4	Vietnam	8,6
5	Saudi Arabia	6,4
6	United Arab Emirates	5,3
7	Algeria	5,1
8	Philippines	4,8
9	Iraq	4,5
10	Egypt	3,9
11	Singapore	3,9
12	Hong Kong	3,2
13	Canada	3,1
14	Poland	3,1
15	Iran	2,7

The following graph (Figure 2) depicts the global distribution of iron ore production. As can be seen, China, Australia and Brazil are the largest producers of iron ore worldwide and they supply 60% of all iron ore produced. Iron ore production was 3,000 million tonnes in 2012 and decreased slightly in 2013 and 2014.

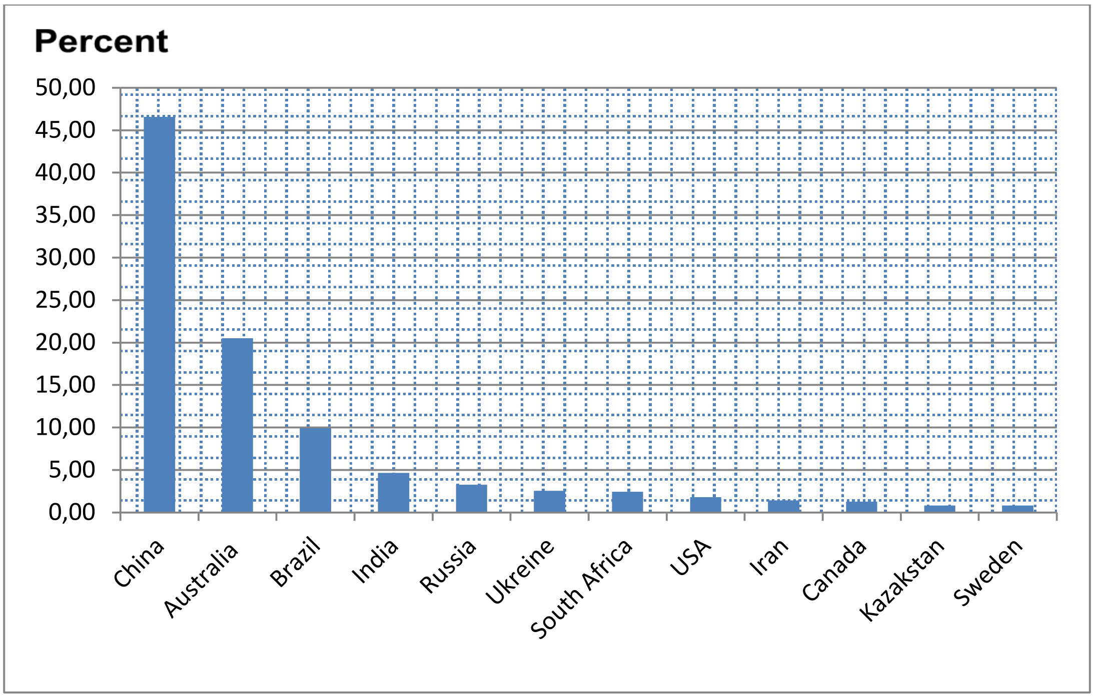


Figure 2 The global distribution of iron ore production (2014) /2/

2.1.1 Iron Ore Resources Worldwide

Sedimentary iron ore deposits are the most important iron ore resource in the world. These deposits are related to the Precambrian. The sedimentary iron ore deposits (Banded iron formations) are the source rocks for most of the large, high-grade concentrations of iron ore currently mined throughout the world. Banded iron formations consist of alternating layers of iron oxide (magnetite and hematite), chert, and a variety of silicates,. They are found throughout the world and are the most important source of iron ore today. Their formation is not fully understood, though it is known that they formed by chemical precipitation of iron oxides from shallow seas during the Archean and Proterozoic.

Iron ore mines are found in at least 50 countries around the world and found in large numbers in the US, Canada, Australia, China, Brazil and India.

World resources are estimated to exceed 800 billion tonnes of crude ore containing more than 230 billion tonnes of iron. The world's iron ore reserves are estimated at

170 billion metric tonnes with iron content of 81 billion metric tonnes. Table 2 shows the amount of iron ore reserves in some countries.

Table 2 World´s iron ore reserves and iron content /4/

	Reserves(million Tonnes) Crude ore	**Iron content**
United States	6,900	2,100
Australia	35,000	17,000
Brazil	31,000	16,000
Canada	6,300	2,300
China	23,000	7,200
India	8,100	5,200
Iran	2,500	1,400
Kazakhstan	2,500	3,300
Russia	25,000	14,000
South Africa	1,000	650
Sweden	3,500	2,200
Ukraine	6,500	2,300
Venezuela	4,000	2,400
Other countries	14,000	7,100
World total (rounded)	**170,000**	**81,000**

2.1.2 World Production

The demand for iron ore has increased sharply over the past few years, by the rapid economic development. In 2014 global output reached 3.22 million tonnes, an increase of just 3.5 % when compared to last year.

China is a world leader in iron ore production. Its share in 2013 reached 46% and in 2014, 47% of world total. China is the largest importer of iron ore in the world, accounting for about a third of the global market as well /3//4/.

Construction and manufacturing industry and automotive production in china caused to increase the iron ore production and growth of the steel industry in recent years. The government reduced the resource tax on iron ore for those vertically-integrated entities involved in both mining and metallurgical processing. The tax reduction was in line with the government's policy to promote integrated iron and steel operations, balance the tax burden among different enterprises and encourage competition.

World crude steel production for the 64 countries reporting to the World Steel Association was 1,580 million metric tonnes in 2013 /4/. This is 12.8% higher than in 2010 and China's crude steel production in 2013 reached 783 million tonnes, an increase of 24.3 % on 2010.

Based on statistics compiled by the U.S. Geological Survey, 2014 global iron ore production was 3,220 million tonnes in total (Figure 3), with China leading the industry as both the world's largest producer and consumer of the ore. Production from China in 2013 was roughly 1,500 million tonnes, or approximately 46.5% of world production. Major iron ore producer countries and their production amounts in recent years are listed in Table 3.

Table 3 Iron ore production amount in major producer countries /3//4/

	Iron ore production(Mt)						
	2008	2009	2010	2011	2012	2013	2014
Australia	342	394	433	480	525	609	660
Brazil	355	300	370	390	375	317	320
Canada	31	32	37	37	40	43	41
China	824	880	1070	1200	1300	1450	1500
India	220	245	230	240	245	150	150
Iran	32	33	33	34	37	40	45
Kazakhstan	23	22	24	24	25	26	26
Mauritania	11	10	11	11	12	11	11
Mexico	12	12	14	14	13	13	13
Russia	100	92	101	100	100	105	105
South Africa	49	55	59	55	61	72	78
Sweden	24	18	25	25	25	26	26
Ukraine	73	66	78	80	81	82	82
United States	54	27	50	54	53	53	58
Venezuela	21	15	14	16	20	20	20
Other countries	47	43	48	50	61	83	85
World total	2220	2240	2590	2800	3000	3110	3220

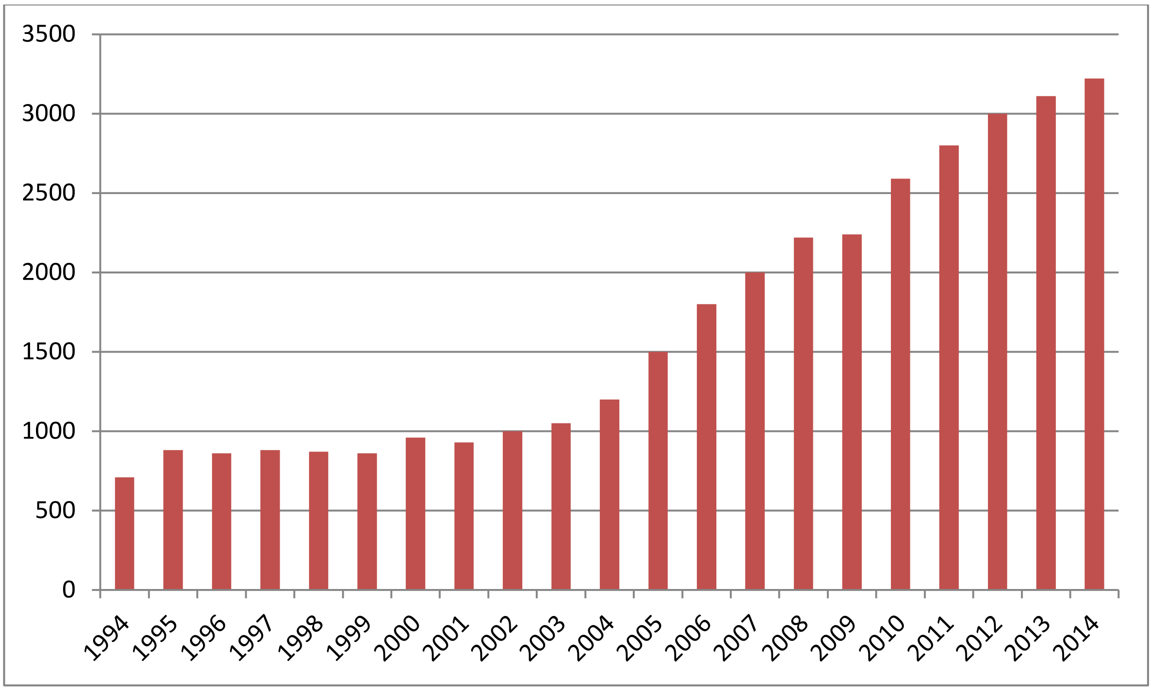


Figure 3 Worldwide iron ore production from 1994 to 2014 (million tonnes) /4/

Among the largest iron ore producing nations are Brazil, China, Australia, India, Russia and Ukraine. Worldwide, 50 countries produce iron ore, but 96% of this ore is produced by only 15 of those countries. Plant capacity of the world's largest iron ore producer groups or companies in 2015 are listed in Table 4.

Table 4 World's largest iron ore producers in 2015 /19/

Company	Base	Capacity Mt/y
Vale Group	Brazil	523
Rio Tinto Group	UK	464
BHP Billiton Group	Australia	395
Fortescue Metals Group	Australia	96
ArcelorMittal Group	UK	82
AnBen Group	China	77
Anglo American Group	South Africa	62
Metalloinvest Group	Russia	48
Evrazholding Group	Russia	48
LKAB Group	Sweden	48
Metinvest Holding Group	Ukraine	48
Cliffs Natural Resources	USA	41
CVG Group	Venezuela	41
Shougang Beijing Group	China	40
NMDC Group	India	36
Imidro Group	Iran	36
CSN Group	Brazil	30
Atlas Iron	Australia	28
US Steel Group	USA	25
Poltavsky	Ukraine	25
Total capacity		**2193**

2.2 Iron Ore in Iran

Iran produces 38 million tonnes of iron ore per year. The major producers in Iran are public corporations, including Chadormalu, Golgohar, Choghart and Sangan, which account for 90% of total iron ore production. The remainder is produced by the private sector including plants in Khorasan (Sangan and Khaaf), Kerman, Yazd, Zanjan and East Azerbaijan. The domestic production of steel in Iran nearly equals the domestic level of iron ore production. Zob Ahan Isfahan, Mobarakeh Steel and Hormozgan Steel are among the largest consumers of iron ore and producers of steel in Iran.

Iranian iron ore export is mainly performed by private companies and almost 90% is exported to China and India.

The domestic consumption of steel in Iran is 18 million tonnes and its per-capita consumption is reported to be 264 kg per capita and as stated in chapters 4.2 and 4.3 of this report it is predicted to increase.

2.2.1 Resources

Inferred mineral resources of iron ore in Iran were estimated at around 11 billion tonnes, which is equal to 3.7% of inferred mineral resources in the world and proved ore reserves were estimated at 2.5 billion tonnes, which is equal to 1.5% of the proved iron ore reserves in the world (Table 2).

Figure 4 The main iron ore regions in Iran

Generally, iron ore deposits in Iran are located in four regions (Figure 4).

(1) Bafgh Area: Ghoghart mine, Chadormalu mine, Chahgaz mine, Sechahoon mine, North anomaly,…

(2) Sirjan Area: Golgohar mines and deposits

(3) Zarand Area: Jalalabad mines

(4) Northeastern Area: Sangan mines

The mine or deposit, amount of reserves and iron grade in each area are shown in Table 5. It is necessary to mention that all iron ore mines in Iran are extracting by open pit mining, and there is no underground iron ore mining experience in Iran.

Table 5 Iran major iron ore mines and deposits /25/

Region	Mine or Deposit	Proved Reserves (MT)	Fe (%)
Bafgh Area	Chadormalu	320	55
	Choghart	93	57
	Sechahoon	106	47
	Chahgaz	81	53
	Mishdovan	50	53
Sirjan Area	Golgohar 1	120	55
	Golgohar 2	47	54
	Golgohar 3	600	54
	Golgohar 4	90	54
Zarand Area	Jalalabad	140	53
Northeastern Area	Sangan 1 (West)	440	43
	Sangan 2 (Centre)	190	44

2.2.2 Production

Golgohar 6 iron ore deposit and other deposits and mines in the Golgohar area belong to the IMIDRO Group. Iran's total iron ore production was 37.8 million tonnes in 2013 and iron ore production in the Golgohar area was 7.9 million tonnes in that year. Table 6 shows the amount of lumps and concentrate of iron ore in large iron ore mines in Iran. The increase of iron ore production in Iran from 2002 until 2013 is shown in Figure 5.

Table 6 Iranian iron ore production in 2013 (million tonnes) /25/

	Lumps	Concentrate	Total (MT)
Chadormalu	1	8.5	9.5
Bafgh Area mines	3.7	6.6	10.6
Golgohar 1, 2 & 4	1.5	6.4	7.9
Sangan mines	0.6	4.5	5.1
Jalalabad	0.3	3	4
Others	1.7	0	1.7
Total	8.8	29	37.8

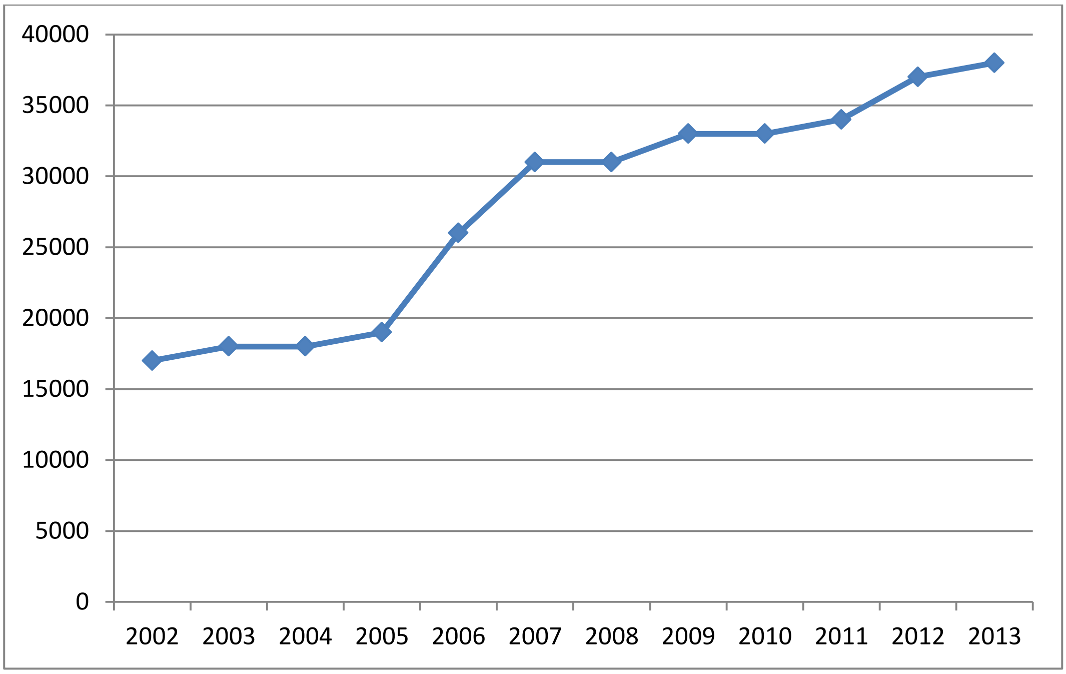


Figure 5 Iron ore production in Iran by year (thousand tonnes) /30/

3 Necessity of Increasing World Production

Considering the growth of the world's population and the increase of urban populations, it is predicted that steel consumption will experience an increase. The level of increase in developing countries is much higher. As observed in Table 7, the per capita consumption of steel in developing countries is significantly high. Considering the relatively high population density in developing countries such as China, India, Turkey, Brazil and Iran, the increase of future production and consumption seems definite.

Table 7 Top ten countries in steel consumption in 2013 per capita

Item	Country	Consumption (kg/capita)	Item	Country	Consumption (kg/capita)
1	South Korea	1133	6	Canada	403
2	Japan	506	7	United State	305
3	China	488	8	Russia	294
4	Germany	449	9	Iran	264
5	Italy	410	10	India	57

3.1 Consumption

The international market of iron ore lacked supply in early 2000 because of the rising demand for steel in developed and developing countries. The rising consumption was driven by the growth of automotive industry in line with the increase in income of the people.

The global economy is growing and especially the middle class in developing countries is growing rapidly. One essential ingredient of this trend is steel and, therefore, the demand for iron ore is growing.

In the past several years, China has continued to be the largest consumer of steel in the world, consuming around 2/3 of total production. Besides China, Japan also needs steel in a considerable quantity to rehabilitate development in the county that has been devastated by the powerful earthquake and tsunami.

In recent years, the international iron ore prices have been on a very high level, resulting in the demand for iron ore demand being seriously distorted and the world's iron ore production totalled 3,000 million metric tonnes in 2013.

3.1.1 Growth of the World Population

The slope of world population growth was gentle until 1650 and afterwards increased swiftly. Figure 6 shows variations in the growth rate of world population.

Rapid population growth is created under the influence of different economic, social and cultural factors. The peak of population growth is observed along with some changes such as the improvement of health, food, literacy and other factors associated with the reduction of mortality rate. Consequently, population, especially in developing countries, has increased significantly. Factors such as the Industrial Revolution in the 18th century, expansion of America and emigration of Europeans to America are among the important factors that have created drastic changes in the population growth rate.

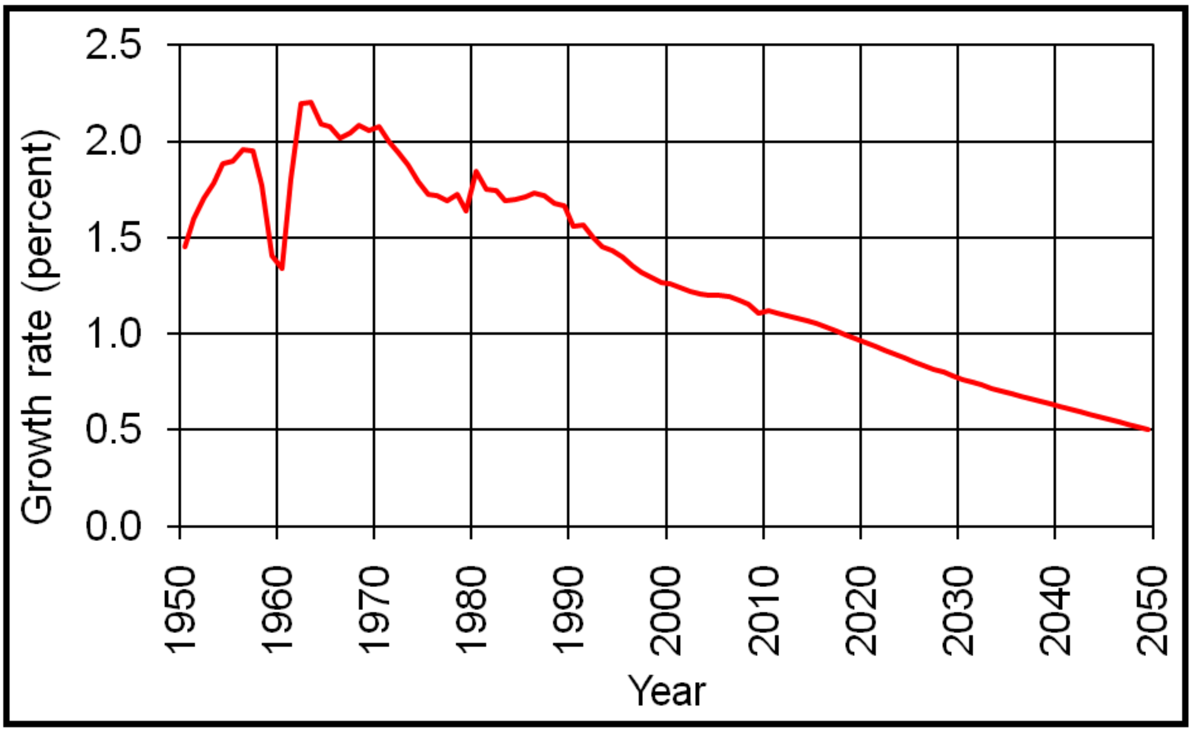


Figure 6 World population growth rates /26/

Currently, developing countries are experiencing ascending population growth, while developed countries have a gentle positive rate and some countries are even experiencing a negative rate. In spite of the reduced birth rate in the world, the world population will still be increasing in the future, especially in developing countries, due to magnitude of the base population, the ratio of young to old population and low mean age. According to recent estimations, world population growth will continue in the next 40 years. In 2050, world population will be greater than 9 billion. This means that steel consumption will also increase.

3.1.2 Urbanization

According to the United Nations Population Fund, about half of the world's population lived in urban areas in 2008. This number is forecast to reach 60% in 2030.

Urbanization is associated with shifts from an agriculture-based economy to mass industry, technology, and services. For the first time ever, the majority of the world's population lives in a city, and this proportion continues to grow. One hundred years ago, two out of every ten people lived in an urban area. By 1990, less than 40% of the global population lived in a city, but as of 2010, more than half of all people live in an urban area. By 2030, six out of every ten people will live in a city, and by 2050, this proportion will increase to seven out of ten people.

Globally, urban growth peaked in the 1950s, with a population expansion of more than 3% per year. Today, the number of urban residents is growing by nearly 60 million every year. The global urban population is expected to grow roughly 1.5% per year between 2025 and 2030. By the middle of the 21st century, the urban population will almost double, increasing from approximately 3.4 billion in 2009 to 6.4 billion in 2050. Almost all urban population growth in the next 30 years will occur in cities of developing countries. Between 1995 and 2005, the urban population of developing countries grew by an average of 1.2 million people per week, or around 165,000 people every day. By the middle of the 21st century, it is estimated that the urban population of these counties will more than double, increasing from 2.5 billion in 2009 to almost 5.2 billion in 2050. Nonetheless, on average, the rate of urban population growth is slowing in developing countries, from an annual rate of roughly 4% from

1950 to 1975 to a projected 1.55% per year from 2025 to 2050. In high-income countries, on the other hand, the urban population is expected to remain largely unchanged over the next two decades, increasing from 920 million people to just over 1 billion by 2025. In these countries, immigration (legal and illegal) will account for more than two-thirds of urban growth. Without immigration, the urban population in these countries would most likely decline or remain static.

In 2014, cities with fewer than 500,000 inhabitants accounted for about half of the world urban population, amounting to 1.94 billion (Figure 7).

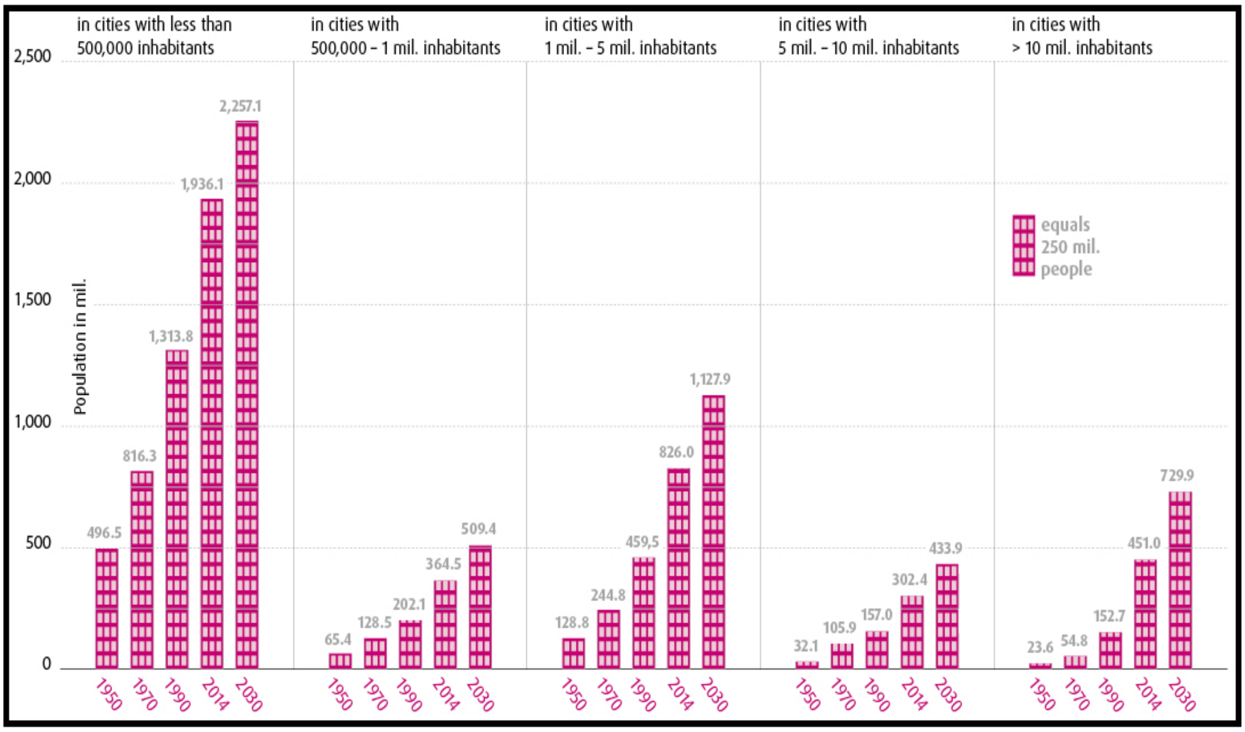


Figure 7 Number of inhabitants in cities in 1950,1970, 1990, 2014 and 2030 /27/

Based on Department of Economic and Social Affairs of United Nations report, 50 percent of world population were living in urban areas in 2005 and increased to 52.8 percent in 2014. There is expected to be over 80 percent urban by 2050.

On the other hand, based on Figure 8 , the consumption of steel is mainly related to construction, mechanical products and metal goods, automotive and transport industries, so the consumption of metal in these items will increase with urbanization.

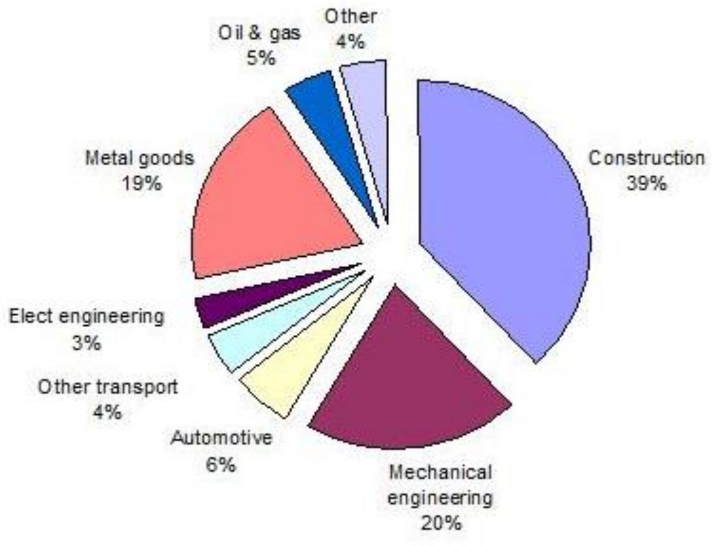


Figure 8 World steel demand analysis by end-use industry /28/

3.2 Iron Ore Prices

Iron ore demand of China caused to increase the iron ore price in 2008, but declined in 2009 due to the economic crisis. After that, there was some fluctuation in the price of iron ore (Figure 9). A comparison of iron ore prices with that of other metals and indexes such as S&P500, Dow Jones or DAX shows that the average price of iron ore is expected to increase over the next years.

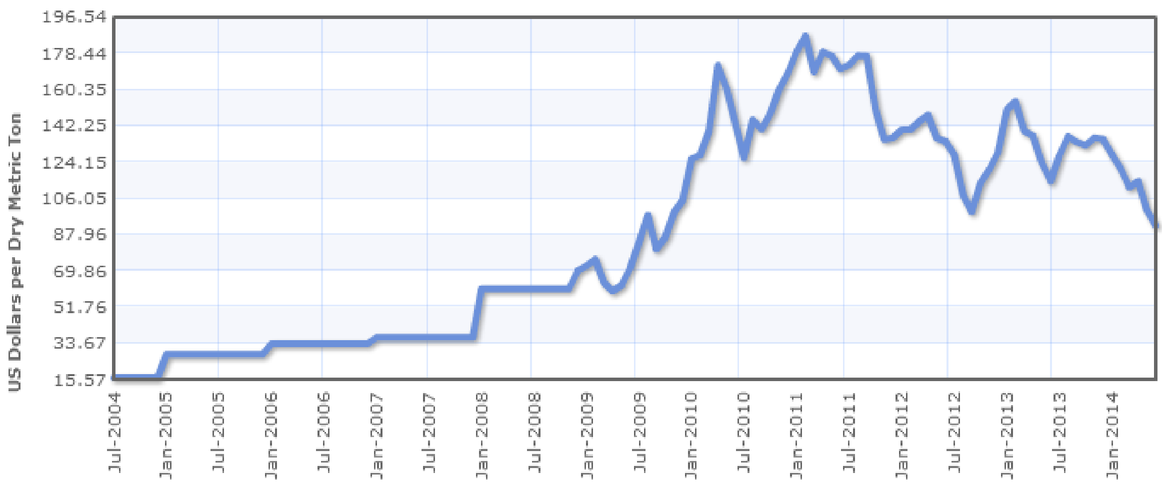


Figure 9 World iron ore prices /29/

3.3 New Extraction Technology

The new extraction technology has led to a reduction in production costs and an increase in productivity. This technology has also reduced environmental pollution and negative impact on human health.

Nowadays, in open pit mine transportation, dump trucks with a capacity of more than 400 tonnes are used or IPCC (In-Pit Crushing and Conveying) method is economic, efficient and environmentally friendly way, and in underground mines (e.g. in sublevels or block caving and room & pillar methods), the LHDs have a capacity of more than 20 tonnes.

A reduction of operating costs has led to the consideration of the possibility of extracting low grade ore and deep deposits.

4 Necessity of Increasing Production in Iran

According to what was mentioned above, it could be concluded that the increase in iron ore production is mainly rooted in the low cost of energy, the relatively low cost of labour, the increase in demand for steel due to population growth, the increase of urbanization, the development of industries and the development of new steel plants, which are all explained in the following section. Thus, according to below reasons, assessment and study of new iron ore deposit is necessary;

(a) As shown in Figure 10, iron ore consumption will increase in future years and this needs an increasing amount of extracted iron ore. This will be possible by starting and developing deeper iron ore deposits and/or the extraction of lower grade iron ore deposits.

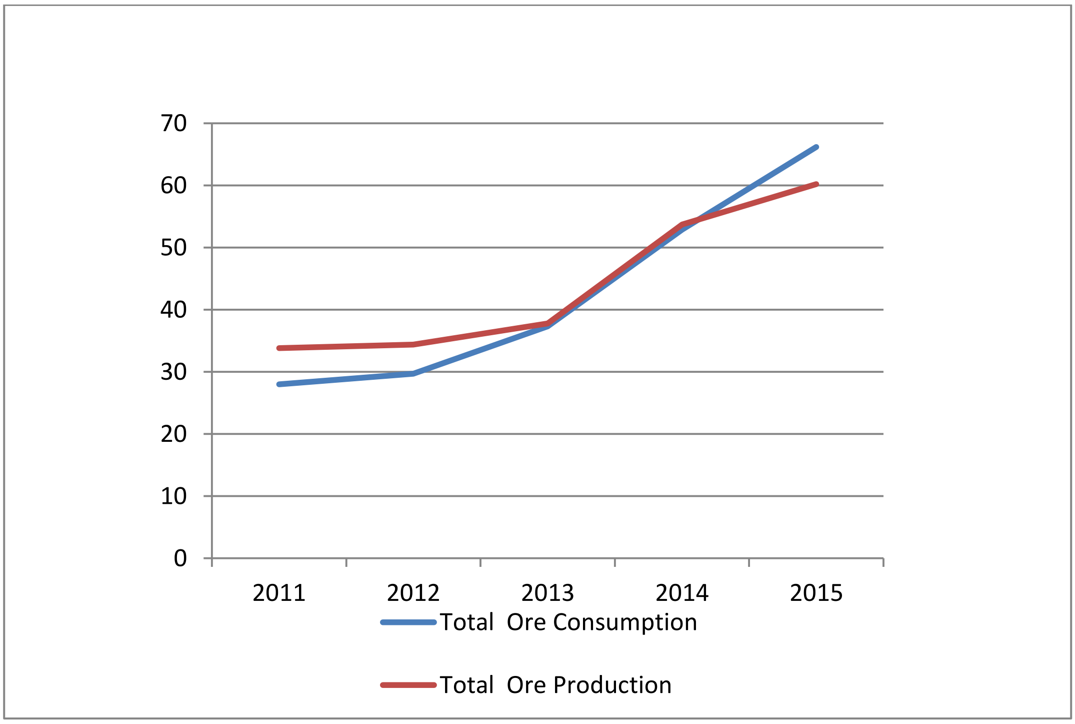


Figure 10 Iron ore production and consumption in Iran (million tonnes) /30/

(b) As can be seen in Table 8 and Figure 10, increasing steel production has been considered by the Iranian Ministry of Mining. The IMIDRO Group is constructing a few steel factories and trying to increase its production of steel by a coal-based direct process, electric arc furnace and rotary kiln process methods. Therefore, the procurement of iron ore for new factories from a relatively close distance is obligatory. According to Figure 10, the production of iron ore in 2014 will be approximately equal to the consumption of iron ore, but in 2015, the production will be less than the consumption of iron ore (about 6 million tonnes). Therefore, the extraction of iron ore in other deposits in preparation of the deficiency of iron ore is necessary.

Table 8 Iron oxide minerals production in Iran /25/

Year	2011	2012	2013	2014	2015
Iron ore lamps consumption	7.1	7.3	9.4	12.1	14
Iron ore lamps production	13.1	11.7	8.8	8.8	8.8
Iron ore concentrate consumption	20.9	22.4	27.9	40.8	52.2
Iron ore concentrate production	20.7	22.7	29	44.9	51.4
Total iron ore consumption	28	29.7	37.3	52.9	66.2
Total iron ore production	33.8	34.4	37.8	53.7	60.2
Deficiency or surplus value	5.8	4.7	0.5	0.8	-6

(c) Depletion of open pit mines and near surface reserves has led to the start of the exploitation of deeper reserves (by underground mining or surface mining).

(d) A sensitivity analysis of iron ore price fluctuation for the coming years shows that iron ore production has an economic safety margin. Likewise, the result of this report shows a sufficient economic safety margin for iron ore extraction.

(e) The increasing world population, the economic growth in developing countries, the growing middle class population in developing countries and urbanization, especially in China and India, have caused an increase in demand for iron ore and steel. By 2025, one billion people will live in Chinese cities. The Chinese urban population is expected to grow by 350 million. China will have 221 cities with more than one million inhabitants. Currently, Europe has only 25 cities with over one million people.

(f) Modern exploration is a process that operates in stages. In general, each stage of exploration work and feasibility study at a related level is designed to get to the next decision point, i.e. whether or not to continue exploration on a property, based on the results of the previous stage. Each successive stage is, in general, more expensive due to the progressively more detailed nature of the work required. Whenever an exploration programme is carried out to get to the next stage, the value of a property may be enhanced, reduced, or remain the same, depending on how the results of the programme affect the perceived exploration potential. For this assessment, the mining method must be determined.

Iran, as a developing country, has a 4 to 5% GDP growth rate and its per capita GDP is 12800 $. The current population of Iran is 70 million, with a growth rate of 1.3 %, and 50% of its population is less than 35 years old. Urbanization in Iran is increasing at the rate of 3%. Iran possesses many oil and natural gas reserves, the second largest hydrocarbon reserves in the world. Accordingly, the cost of energy is relatively low in Iran.

4.1 Inexpensive Energy

Iran, as the owner of 10% of discovered oil resources and 15% of gas resources in the world, is one of the major international exporters of oil and gas.

Furthermore, the appropriate topographical conditions have provided the ground for the construction of dams or dykes in mountainous regions and the possibility of utilizing hydroelectric plants. Coal mines are another energy source in Iran, which exist in two forms: thermal coal and coke.

Regarding the climate of Iran, the installation and application of solar energy equipment and wind power is possible. Currently, these sources are used in certain areas in Iran.

Considering the inexpensive energy sources and easy accessibility of such sources as well as the energy consumption of steel industries, which is significantly high, it could be argued that inexpensive access to energy sources is one of the advantages for the development of steel production and the increase of iron ore extraction in Iran.

4.2 Development of Population and Cities

One of the reasons for the large demand for steel is population growth. According to published statistics, the population of Iran has doubled in the last 40 years. Figure 11 shows the development of population growth. At the same time, statistics indicate that in 2012, 55% of the population of Iran was less than 30 years of age, which suggests that Iran is a young country. This issue has a two-dimensional significance:

(1) Providing jobs, housing, food, health, education, etc., for the increasing younger population requires further investment and more raw material consumption, such as steel.

(2) The youth population of a country indicates population growth in the future.

In addition to these two factors, urban population is also growing in Iran. For instance, the population of Tehran has increased from two to nine million between 1960 and 2012. Obviously, steel consumption in the cities is higher than in rural areas and, generally, kind of materials that utilities in buildings and municipal facilities such as transportation (subway, bus, etc.) depend on steel consumption. Moreover, infrastructure, such as water, gas, tunnels, bridges, subway lines and tramways, requires large volumes of steel.

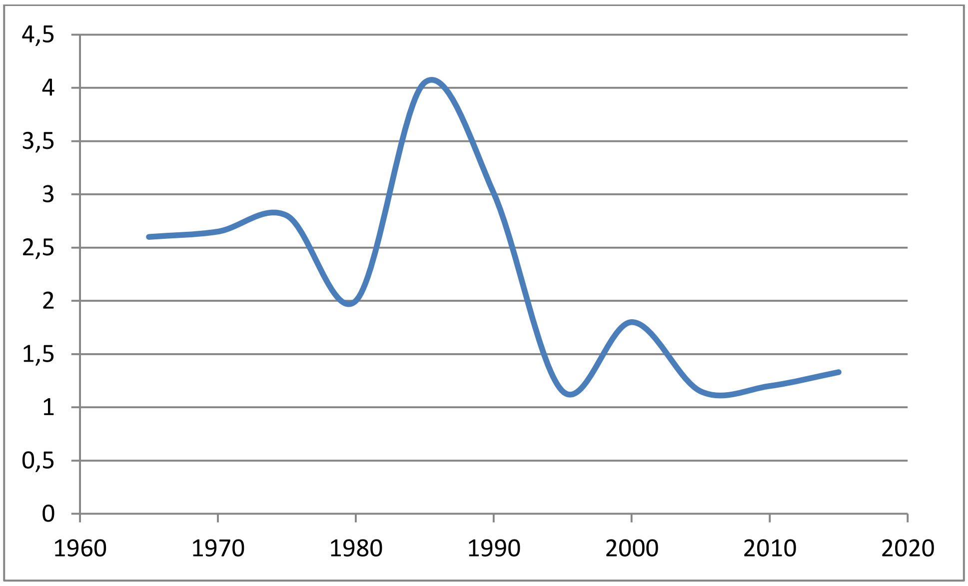


Figure 11 Iran population growth rate per year /31/

4.3 Development of Industry

The development of industry, including the construction, transportation, agricultural industries, etc., requires a higher steel consumption. Such a development has occurred recently in Iran and it is estimated that Iran, as a developing country, will need much steel in order to continue its development. For example, Table 9 indicates the growth of the automobile industry in Iran from 2000 to 2012.

Table 9 Automobile production in Iran since 2000 /32/

Year	*Number(*1000)*	*Year*	*Number(*1000)*
2000	278	2007	997
2001	323	2008	1051
2002	455	2009	1394
2003	582	2010	1599
2004	788	2011	1648
2005	817	2012	989
2006	904		

Further, the level of cement consumption may reveal the industrial growth and the construction of buildings, plants, factories and urban facilities. The amount of cement production in Iran is currently about 60 million tonnes which is 2% of the world production with 3300 million tonnes. The level of cement production in Iran has doubled within the last ten years.

4.4 New Steel Production Factories

As mentioned in Chapter 2, the level of steel and iron ore production has been increasing in Iran and, currently, the annual volume of steel and iron ore production is 15 and 38 million tonnes, respectively.

In addition, some new projects for steel production have been established. This will increase the consumption of iron ore in the near future. The following are some of the new projects:

(1) Bafgh Steel Factory (0.8 MTPA)

(2) Bonab Project (0.6 MTPA)

(3) Natanz Steel Company (1.1 MTPA)

(4) Mobarake and Hormozgan Development Projects (2 MTPA)

After the completion of these projects, the volume of production will reach 10 million tonnes, which suggests an annual increase in demand of almost 25 million tonnes of iron ore.

According to IMIDRO reports, given the increase of steel production plants, the consumption of iron ore as lumps and pellets will rise in the future. Figure 10 shows the iron ore production and consumption in Iran and Table 8 indicates the steel production methods including direct reduction steel production and blast furnace as well as required amount of iron ore.

4.5 The Explored Ore Deposits

Considering the factors behind the increase in steel consumption, including population growth, urbanization, increase in steel consumption per capita and other parameters specific to Iran, including access to cheap energy, the establishment of new iron ore mines is vital. Obviously, new mines have different properties, which could make the extraction operations more cumbersome. In this case, the depth of the ore deposit and ore grade can be noted. At first glance, ore deposits with lower depth and a lower stripping ratio in open pit mining method are much more economical. Along with new developments in mining machinery, the extraction of deeper ore would also be cost-effective. In addition, with the development of machinery and extraction methods as well as mining equipment, the extraction of ore with low grade will be economically possible.

As observed in Table 5, Iranian iron ore mines with a minimum of 43% Fe are extracted using the open pit mining method. Table 10 indicates iron ore deposits which are in the detailed exploration phase.

Table 10 Iron ore deposits in exploration /25/

Item	Deposit	Area	Resource (MT)	Fe%
1	A12 anomaly	Bafgh	80	59
2	North anomaly	Bafgh	232	31
3	Sheitoor	Bafgh	12	52
4	Saghang 20,21	Bafgh	1,100	56
5	D19 anomaly	Bafgh	110	45
6	A13 anomaly	Bafgh	10	52
7	Golgohar 6	Sirjan	90	56
8	Sangan east	Northeastern	410	43

To compare these deposits and choose the most appropriate one for mining, some parameters, including the resource amount, geometrical properties and infrastructure are important. Based on the availability of railway, roads, electricity and other infrastructural necessities, IMIDRO decided to complete exploration of the Golgohar 6 anomaly. Golgohar iron ore 6 deposit is located in southern Iran, 50 km from the city of Sirjan, in the southwest of Kerman Province, surrounded by mountains with a height of over 2,500 m.

The evaluation method and the procedure for selecting the most appropriate mining method are first examined. Afterwards, the Golgohar iron ore 6 anomaly is investigated as a case study for the selected method.

5 New Mines Exploitation

For exploitation of new mines, the method of extraction should be first analysed. The selection of an appropriate extraction method is one of the most difficult processes in mining engineering. The ultimate goal is to maximize the revenue, increase the recovery of reserves, and ensure the maximum safety and environmental protection. The selection of the extraction method depends on different factors, such as technical, economic as well as social and regional parameters. Social and regional parameters include history of extraction, availability of skilled and efficient manpower as well as the presence of infrastructure, such as transportation facilities and energy supply. However, confining environmental factors may play a significant role. The present report only discusses the selection of the extraction method from a technical standpoint and other parameters are not within the scope of the present research.

An appropriate extraction method considers technical factors such as geometry of the deposit as well as ground conditions. This means that the most appropriate extraction method is a method with minimal technical problems during extraction operation. Therefore, usually, there is more than one method for mining an ore deposit. Obviously, each possible method would have specific drawbacks. Finally, a method with minimal technical problems and the best economic conditions will be selected.

Some researches related to mine extraction method selection have been carried out by many scientists, such as Boskhov-Wright, Hartman, Laubscher and Nicholas. Evaluations that are made with classic methods generally produce a complex situation and require a long period. In particular, due to questions on many parameters and uncertain elements in mining method selection, the decision making process is made more difficult.

5.1 Mining Method Selection

Selecting the method used by adjacent mines would not always work. However, an examination of the extraction method used by adjacent mines, which can be considered a potentially appropriate method, is very important. Each ore deposit possesses particular properties that may be completely different from those of others. Technical investigations may help in the selection of the extraction method.

Nowadays, the selection of an extraction method is made on the basis of engineers' judgements and experts' experience. This involves collaboration between geologists and engineers to acquire exact data of the ore deposit to be analysed. Such data are usually related to physical properties, mainly determined based on the results of exploration, including the shape and size of the deposit, chemical analysis, mechanical strength, conditions of hanging wall and foot wall. In addition, technical and economic parameters, including the possibility of mechanizing extraction, recovery rate of the ore, flexibility of the extraction method, selectability of mineral extraction, and capital and cost-related factors. On the other hand, factors such as environmental considerations and subsidence are also important.

In order to determine which mining method is feasible, the characteristics of the deposit need to be compared with those required for each mining method. The method(s) that is/are best match should be the one(s) considered technically feasible, and should then be evaluated economically.

This research discusses the pre-planning stage of mine development. A mineral deposit of sufficient quality to justify mining is assumed. At this point, a decision needs to be made whether the method of mining should be surface or underground. Usually, the depth and size of the deposit makes this decision obvious. However, this is not always the case. The factors affecting the choice of either of the two options are given below.

Generally, the specification of open pit mining such as safety, ore loss and dilution, mechanization, grade control and cut-off grade, economics and production capacity cause to select this method as mine extraction method . The advantages of open pit mining are:/7/.

- Good for shallow deposits. The maximum depth that can be mined by this method is dictated by the technology used. With rapid advances in technology, surface mines have gone significantly deeper than before. For example, the Bingham Canyon copper mine in Utah is about one kilometre deep.
- Generally lower cost per tonne than underground methods. Therefore, even relatively poor grades can be mined as well.

- Very disruptive to the environment. According to a study in 1978, 75% of the land affected by surface mining is due to mining of coal, gravel and crushed stone. Reclamation can be expensive. Sometimes, companies prefer to go underground (despite bad economics) simply to alleviate environmental concerns.

- Mining is affected by weather. Inclement weather can lead to mining stoppages.

- Lighting. Does not require artificial lighting during day hours.

- Multiple seams can be mined without being subject to ground control problems. In Yellandu, India, the Singareni Colleries Company Limited mined seams using surface methods that were previously mined using underground methods.

- High capital required for modern mines.

- Cannot be used for selective mining.

- Generally higher productivity than underground mines.

It should be mentioned here that not all factors mentioned above apply. For example, if a company has several open pit mines, and decides to open a new one, the capital required will be less if they divert some equipment from the existing mines.

Underground mining, however, can be considered as being more acceptable than surface mining from environmental and social perspectives. In addition, underground mining will often have a smaller footprint than an open pit of comparable capacity.

- Good for relatively deep deposits. Usually, the depth is more than 100 feet. In many situations, the depth factor makes the decision simple. If the deposit is too deep, surface methods are ruled out.

- Generally, the cost per tonne is higher than for surface methods. Therefore, it is good for high quality grades only.

- Less disruptive environmentally. In the past, however, reckless underground mining left behind large tracts of subsided land.

- Underground mining is rarely affected by the climate. However, artificial ventilation and lighting is required. In some very deep mines, mine production is affected by heat (due to the depth).
- More hazardous than surface mines. Mining in a coal seam is affected by present/old workings in other seams.
- Return on capital is generally not quick.
- Generally less dilution when mining. Especially good for complex ore bodies where selective mining can be carried out.

The classification system proposed by Boshkov and Wright (1973), was one of the first qualitative mining method extraction classification for underground mining method selection . It uses general descriptions of the ore deposit such as ore dip, strength of the ore, and strength of the walls, ore thickness to identify common methods that have been applied in similar conditions.

Hartman (1987) /34/ developed a flow chart selection process for defining the mining method, based on the geometry of the deposit and the ground conditions of the ore zone. Based on Morrison (1976) classification system, underground mining methods are divided in three categories: rigid pillar support, controlled subsidence, and caving. General definitions of ore width, support type, and strain energy accumulation are used as the criteria for evaluating a mining method. This classification helps to demonstrate the selection continuum, choosing one method over another based on the various combinations of ground conditions (Morrison 1976). The selection of an appropriate mass underground mining method has been presented by Laubscher (1981) /34/. The selection process is based on his rock mass classification system, which adjusts for expected mining effects on the rock mass strength.

Laubscher (1990) later modified the classification to relate his rock mass rating to the hydraulic radius. By including the hydraulic radius, caveability becomes feasible for more competent rock if the area available for undercutting is enough.

The ore geometry and rock mechanic characteristics of the ore zone, footwall and hanging wall were important characters of deposit in Nicholas (1981) /34/ numerical ranking method. Later, Nicholas made some modifications to his selection procedure by introducing a weighting factor. The UBC (University of British Columbia) mining method selection algorithm developed by Miller, Pakalnis and Poulin (1995) is a modification to the Nicholas approach, which places more emphasis on stoping methods, thus better representing typical Canadian mining design practices.

Multiple criteria decision making is used for the systematic assessment and comparison of alternative solutions to a problem according to qualitative and/or quantitative criteria.

Decision making can be defined as a selection process of the best one among the alternative sets in order to achieve a goal, and mostly has an uncertain situation. Additionally, mostly linguistic variables (weak rock, massive ore deposit, etc.) are in question. Since the application of the fuzzy set theory in 1965, these uncertainties in question are easily evaluated in the decision making process.

The approach to the selection of a mining method can be classified into three groups:

- Qualitative methods
- Numerical ranking methods (scoring)
- Decision-making models

5.2 Algorithmic Model Structure

The three abovementioned methods could be applied to determine likely extraction methods. For this purpose, a qualitative method will first be used to determine possible extraction methods. The qualitative method could define the most appropriate extraction methods, and the distinction between possible and impossible extraction methods would be done by a qualitative method. Then, the possible extraction methods are evaluated and investigated by a numerical ranking method and a decision-making model. An assessment model of an appropriate extraction method selection is shown in the flow chart in Figure 12.

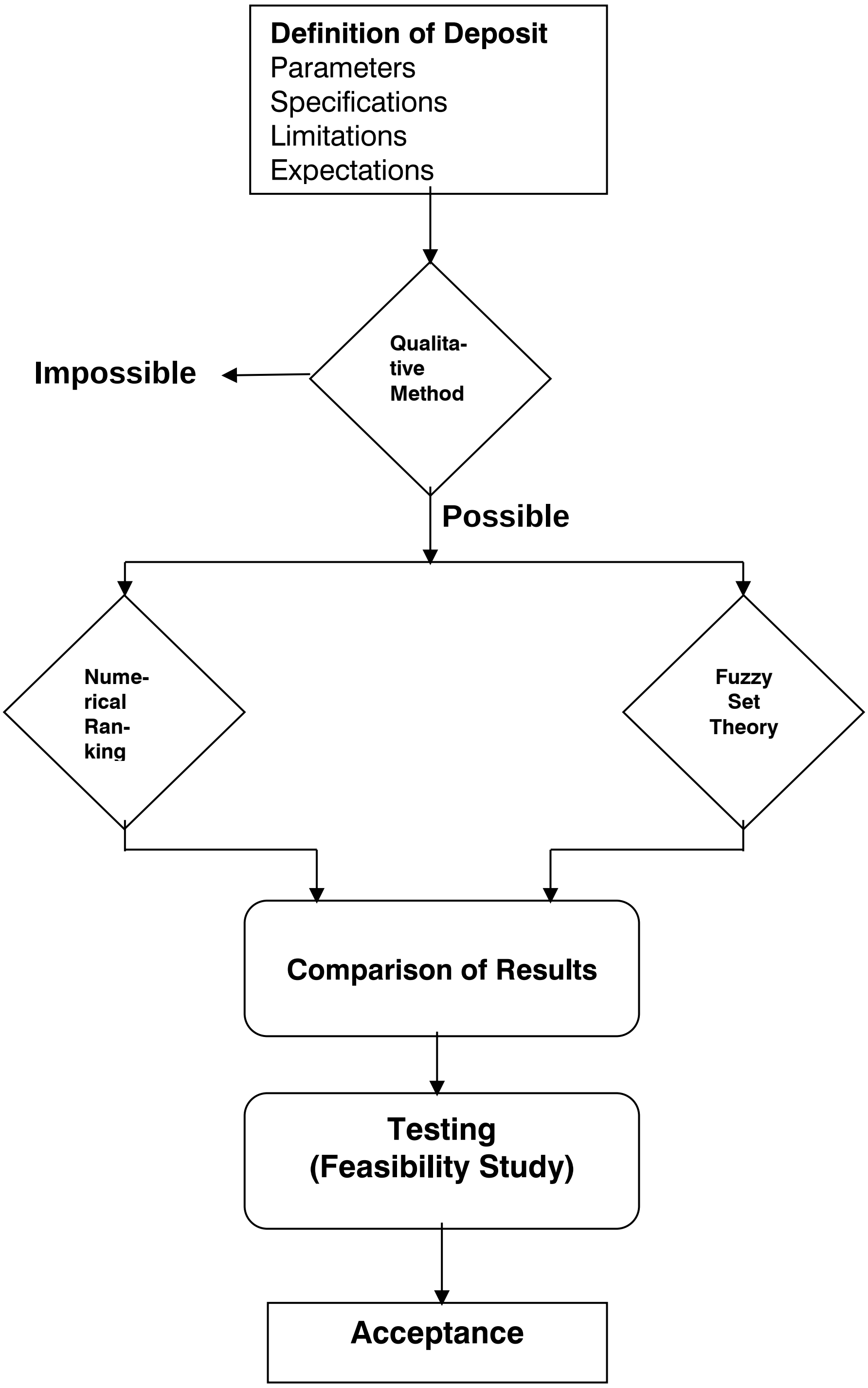


Figure 12 Process sequence modelling

5.3 Qualitative Method

The qualitative selection method has been designed by Hartman (1987) and developed for the classification of traditional mining methods (Figure 13). It is a chart to aid in the selection of the appropriate methods, both surface and underground, and three categories of characteristics, including depth, ore and rock strength, and spatial and geometric considerations, comprise the matrix. The decision-making process proceeds from left to right, successively defining the situation, class and name of the method.

While this selection procedure is simple and quick to use, it is neither definitive nor quantitative. Only relative terms are employed and environmental issues are neglected. Figure 13 is useful as a first approximation and to narrow the choice tentatively to a few candidate methods, but a more selective, quantitative procedure is needed.

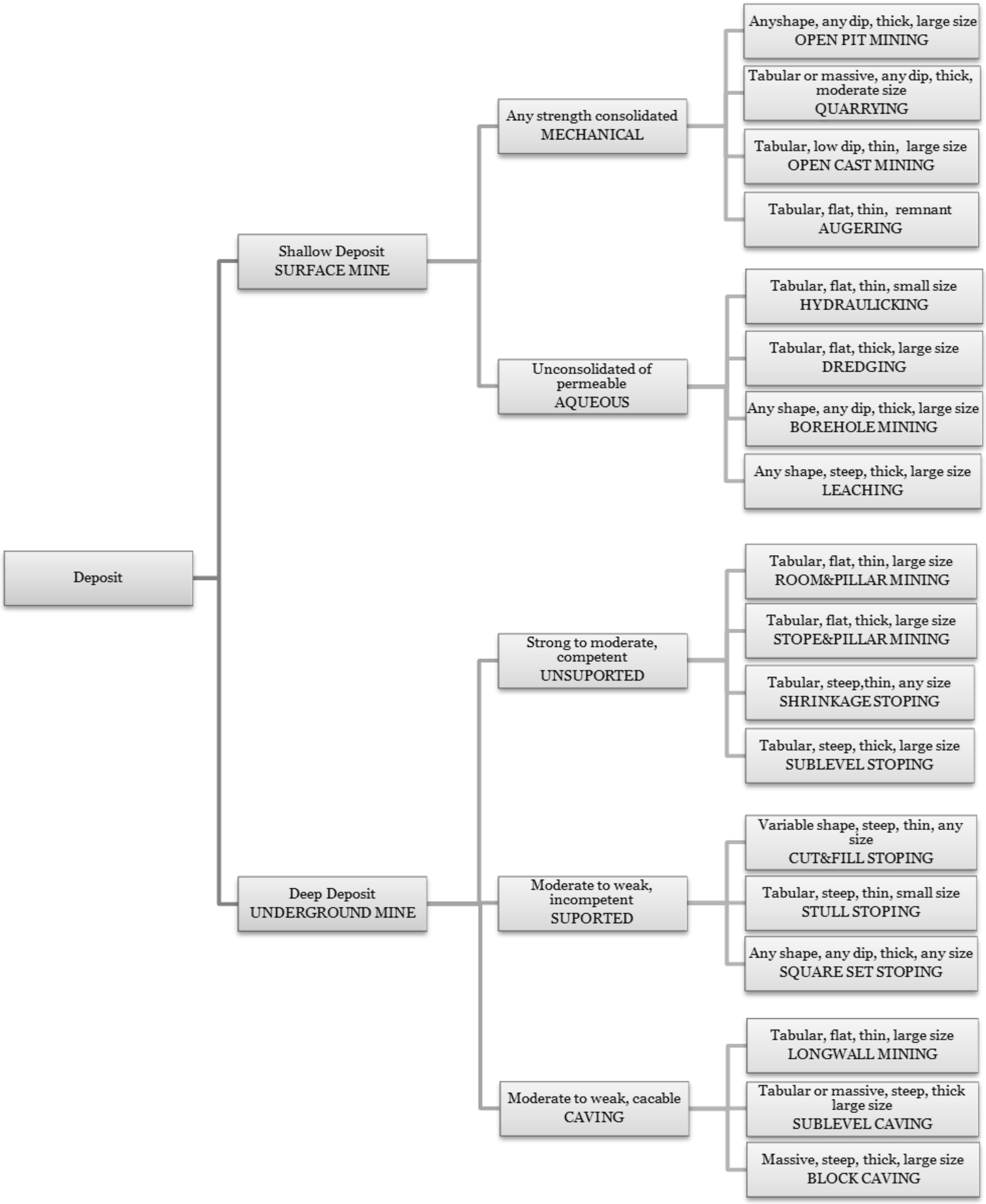


Figure 13 Qualitative selection of an appropriate mining method based on deposit characteristics

5.4 Numerical Ranking Method

To evaluate the most appropriate extraction method, the numerical system is conducted on the basis of weighting the parameters of the ore deposit. After that, the processing of these numerical data results in the ranking of the extraction methods.

Based on ore deposit characteristics such as geomechanical properties of the rock, deposit geometry, depth and thickness,, grade distribution and grade, some qualitative and quantitative extraction method selection have been defined. The selection methods have been presented by Boshkov and Wright (1973), Morrison (1976), Laubscher (1977, 1981, 1990), Nicholas (1981, 1992), Hamrin (1982), Hartman (1987), Miller (1995), Clayton (2002), recently Shahriar (2007) and etc.

One of comprehensive mine extraction method selectin in quantitative methods is defined by Nicholas (1981)., which focused on geometrical and geomechanical specification of ore body, hanging wall and footwall. The Nicholas method is based on a numerical approach for ranking of mining methods based the rate of parameters. A numerical rating for each mining method is arrived at by adding up these rankings. The higher the rating is the more suitable the mining method. Nicholas provides a numerical stage in which mining methods are analysed for suitability with a given mineral deposit. Geomechanical conditions are also specified for the ore deposit and adjacent wall rock.

5.5 Decision-Making Model Based on Fuzzy Set Theory

Acquiring the information necessary for mining method selection is an elaborate process, to say the least, and once obtained, the data is likely to be ambiguous. In addition, decision makers must often apply rules of thumb or incorporate their personal intuition and judgement when deriving performance measures based on indefinite linguistic concepts, e.g. ‘high’, ‘low’, ‘strong’, ‘weak’, and ‘stable’. Such terminology is common and is caused by imperfectly defined problem attributes /9/.

The fuzzy set theory is used to describe fuzzy sets, and was developed as an alternative to ordinary set theory. Fuzzy logic is used to derive the set membership function for a fuzzy set, which is used for fuzzy logic decision making. The problem of constructing meaningful and suitable membership functions involves plenty of additional research. A number of empirical ways to establish membership functions for fuzzy sets are known.

A decision-making model based on fuzzy set theory procedure is also done as a waterfall model (Figure 14).

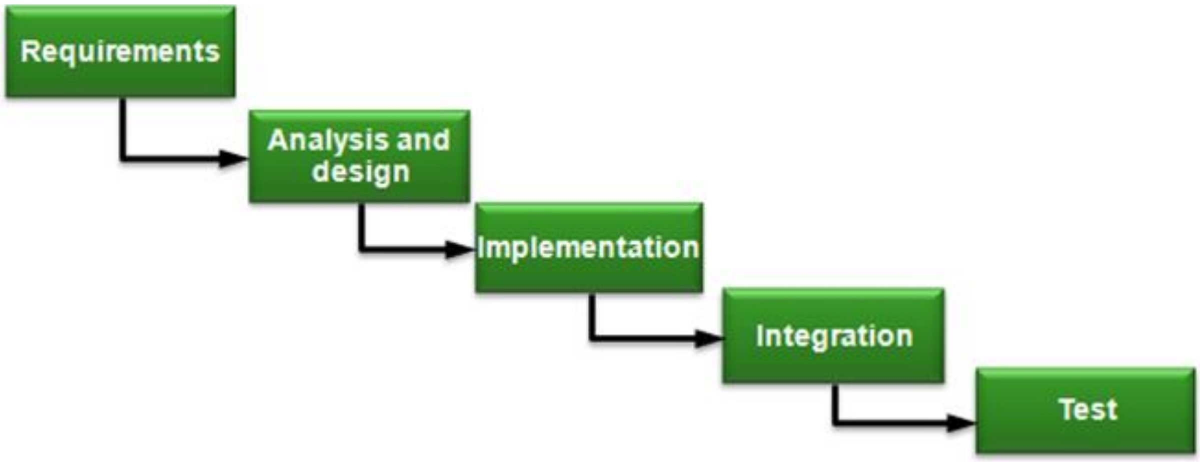


Figure 14 Waterfall model diagram

Schematically, the method of preliminary data acquisition is a multi-dimensional method, which handles, processes and finally obtains the results from the analysis. In the first stage, the method converts the multi-dimensional data into two-dimensional matrixes. Afterwards, the data are constructed as columns to give the results.

5.5.1 Alternatives

The required physical parameters such as geological and geotechnical properties of the ore, hanging and foot wall, economic effects, environmental effects, which are established using field and laboratory tests, were determined together with other uncertain variables. Then, the fuzzy set theory was applied to these parameters, considering the available mining methods in order to choose the proper method.

The alternatives in mining method selection are:

- Room and pillar
- Cut and fill
- Longwall

- Shrinkage stoping
- Block caving
- Hydraulic mining
- Open pit mining
- Sublevel stoping
- Sublevel caving
- Solution mining
- Vertical crater retreat

5.5.2 Determining Evaluation Criteria

There are too many criteria that play a role in mining method selection. In this selection, some of the criteria such as geological and geotechnical properties, economic parameters and geographical factors are involved.

The most important criteria are: RMR (Rock Mass Rating) of hanging wall, RMR of footwall, RMR of ore, depth, deposit dip, deposit thickness, deposit shape, mechanization, technology, ventilation, underground water, expert labour (miners), subsidence, deposit size, investment, recovery, production, ore uniformity, cost, health/safety, environmental impacts, stability, selectivity, dilution, flexibility, ore grade, etc. In view of changes in conditions from one part of a mine to another, it is very difficult to formulate definite criteria for the mining method selection that satisfy all the conditions of the mine simultaneously. Therefore, it seems clear that only an experienced engineer, who has improved his experience by working in several mines and gaining skills in different methods, can make a logical decision about mining method selection.

This large number of criteria leads to computational difficulty, a time-consuming process and an unrealistic outcome. To overcome these problems, main criteria or factors for a suitable mining method selection must be identified.

The objective of this survey was to assess the importance of the abovementioned factors as criteria to be incorporated in the Technique for Order Preference by Similarity to Ideal Solution (TOPSIS) model for the selection of a mining method.

5.5.3 Analytical Hierarchy Process

The analytical hierarchy process that was developed by Thomas Saaty (1988) /24/ has been used to solve the problems of decision-makers in different areas such as politics, defence, town planning, communication and psychology. This process consists of the following stages (Albayrak 1997) /9/:

- Problem is clearly described
- Possible targets are determined
- Factors that affect the targets are determined
- According to the alternatives, the results of the model are analysed

If a set of *n* criteria should be compared pairwise according to their relative weights, where the criteria are denoted by $a_1, a_2, \ldots, a_n$.

$$P = \begin{vmatrix} a_{11} & \ldots & a_{1j} & \ldots & a_{1n} \\ a_{i1} & \ldots & a_{ij} & \ldots & a_{in} \\ a_{n1} & \ldots & a_{nj} & \ldots & a_{nn} \end{vmatrix}$$

where $a_{ji} = 1/a_{ij}$

Matrix *P* was defined the importance of parameters based on experienced engineers estimation /9/. The judgement scale used here is:

1. Equally important
3. Weakly more important
5. Strongly more important
7. Demonstrably more important
9. Absolutely more important

The pairwise comparison of each criterion can be described as shown in Table 11 .

Table 11 The pairwise comparison of each criterion

Quality Criteria	Criterion 1	Criterion 2	Criterion 3	Criterion 4
Criterion 1	1	3	1/5	7
Criterion 2	1/3	1	1/7	2
Criterion 3	5	7	1	9
Criterion 4	1/7	1/2	1/9	1

Based on the pairwise comparison matrix, *P*, the principal eigenvector is computed and normalized /11/. The relative weights are determined from *P* by dividing the elements of each column by the sum of the elements of the same column /10//12/. The geometric means of the i_{th} row, called M_i, is calculated as:

$$M_i=\prod_{J=1}^{n} a_{ij} \quad \text{for} \quad i=1,2,3,....,n$$

where a_{ij} is the element in the comparison matrix *P* standing for the comparison of the i_{th} to the j_{th} criterion . The desired relative weights are then computed as the row average of the resulting normalized matrix. Table 23 shows the amount of M_i, W_i and b_i /12/.

$$W_i=(M_i)^{1/n}$$

$$Bi = \frac{\mathrm{Wi}}{\sum_{i=1}^{n} \mathrm{Wi}}$$

The relative weights of the criteria are finally achieved in the eigenvector /9/,/11/ of the matrix, i.e. eigenvector = {B1,B2,B3,….,Bn}.

The decision-maker is required to make pairwise comparisons between decision alternatives and criteria using a ratio scale. It allows the decision maker to focus on the comparison of just two alternatives, which makes the observation as free as possible from extraneous influences.

Let A = {A1, A2, ..., Am} be the set of possible alternatives and P = {P1, P2, ..., Pn} the set of selection criteria. A decision-maker is asked to define the membership grade of each criterion (Matrix G) after conferring with experts on this subject. Table 12 shows the membership levels of each criterion.

Table 12 The membership level of each criterion (Matrix G)

	P1	**P2**	**P3**	**"**	**Pn**
A1	G11	G12	G13	"	G1n
A2	G21	G22	G23	"	G2n
A3	G31	G32	G33	"	G3n
"	"	"	"	"	"
Am	Gm1	Gm2	Gm3	"	Gmn

Then, the applicable membership decision functions of alternatives A1, A2, A3, ..., Am, respectively, can be defined as follows /13/:

μD(A1) = {(G11)^B1, (G21)^B2, (G31)^B3,...., (Gn1)^Bn}

μD(A2) = {(G12)^B1, (G22)^B2, (G32)^B3,...., (Gn2)^Bn}

…….

…….

μD(Am) = {(G1m)^B1, (G2m)^B2, (G3m)^B3,...., (Gnm)^Bn}

The results of relative weights of the criteria are shown in Table 13 (Matrix R).

Table 13 Weights of the criteria (Matrix R)

$$R = \begin{vmatrix} (G11)^{B1}, (G12)^{B1}, (G13)^{B1}, \ldots\ldots\ldots\ldots, (G1m)^{B1} \\ (G21)^{B2}, (G22)^{B2}, (G23)^{B2}, \ldots\ldots\ldots\ldots, (G2m)^{B2} \\ \ldots\ldots \\ \ldots\ldots \\ (Gn1)^{Bn}, (Gn2)^{Bn}, (Gn3)^{Bn}, \ldots\ldots\ldots\ldots, (Gnm)^{Bn} \end{vmatrix}$$

5.5.4 Ranking the Alternatives

Fuzzy logic is used to derive the set membership function for a fuzzy set, which is used for fuzzy logic decision making. A number of empirical ways to establish membership functions for fuzzy sets are known. In this case, mining method selection can be justified in two ways, justification using the max-min composition method, and justification using priority comparison method.

5.5.4.1 Justification Using the Max-Min Composition Method

The max-min composition method is based on Lotfi Zadeh (1965) /22/ opinion in fuzzy set theory. There is interaction between criteria and parameters, and the final decision will be explicated by a decision function. The decision function is shown below.

$$\mu D(j) = \min \{(G1j)^{B1}, (G2j)^{B2}, (G3j)^{B3}, \ldots\ldots, (Gnj)^{Bn}\} \text{ for all } j = 1 \text{ to } m$$

The appropriate criteria are specified by

$$\mu D(j^*) = \max \mu D(j) \text{ where } j^* \text{ is the optimal decision}$$

Therefore, the minimum relative weight of criteria for each alternative and the optimal solution, corresponding to the maximum membership $\mu D(j^*)$ determines the preferable method /14/.

5.5.4.2 Justification Using Priority Comparison

This method is based on comparison of criteria weight. In the priority comparison technique, the elements of each level are compared to their related element on the upper level. The weights of criteria are elements of Matrix R that is represented below. The comparison of these elements is defined by

$$d_{ij} = \sum_{i=1}^{m} \{[Rij - Ri(j+k)] \geq 0\}$$

where Matrix R is:

$$R = \begin{vmatrix} R11, R12, R13, \ldots\ldots\ldots, R1m \\ R21, R22, R23, \ldots\ldots\ldots, R2m \\ \ldots\ldots \\ \ldots\ldots \\ Rn1, Rn2, Rn3, \ldots\ldots\ldots, Rnm \end{vmatrix}$$

The comparison results (d_{ij}) are placed in Matrix D, where d_{ij} is the priority of element *i* compared to element *j*. Finally, the sum of the matrix columns indicate the appropriate alternative.

$$D = \begin{vmatrix} d11, d12, \ldots\ldots\ldots d1m \\ d21, d22, \ldots\ldots\ldots d2m \\ \ldots\ldots\ldots \\ \\ dm1, dm2, \ldots\ldots\ldots dmm \end{vmatrix}$$

6 Evaluation of Golgohar 6

The Golgohar iron ore complex (including six areas) is located approximately 60 km southwest of Sirjan city, in the Kerman province of Iran (Figure 15) and is accessible by road and railway. Golgohar is situated in the southern Zagros Mountains and in the eastern edge of the Sanandaj-Sirjan structural zone of Iran. The elevation is about 1,720 m, with a semi-desert climate: relatively dry and hot in summer, cold in winter and rainfall in spring.
In 1969, the Golgohar ore deposit was discovered by Iran Barite Co. and was appropriated by the National Iranian Steel Company in 1974. Golgohar follows a mining tradition in this region, which dates back 900 years. The Golgohar mine includes six ore bodies (Figure 17) that are spread over an area of 40 km².

6.1 Summary of Available Information

The following sections provide a summary of available information on Golgohar 6. This information consists of accessibility, geographical and climate and infrastructure situation, geological information, topographical and geological maps and exploration results (airborne and surface geophysical measurements, exploration drilling and core logging, sampling and analysing).

6.1.1 Geographical Conditions, Accessibility and Infrastructure

Golgohar 6 is located between longitude 55° 15` to 55° 24` and latitude 29° 03` to 29° 07` and is connected to Sirjan via asphalt road (Figure 15). A brief statistical guide to the Golgohar area:

- Height above sea level: 1,750 m
- Average annual rainfall: 120 mm
- Highest temperature recorded: + 40°C
- Lowest temperature recorded: - 16°C

- Max. wind speed: 65 km/h
- Average humidity: 30%

Figure 15   Location of Golgohar mine and connected roads

The necessary and infrastructural items such as a railway connection, asphalt road, electrical power, telecommunication and internet connection, living facilities and water exist. Figure 16 presents some climate data and daylight hours of Kerman Province.

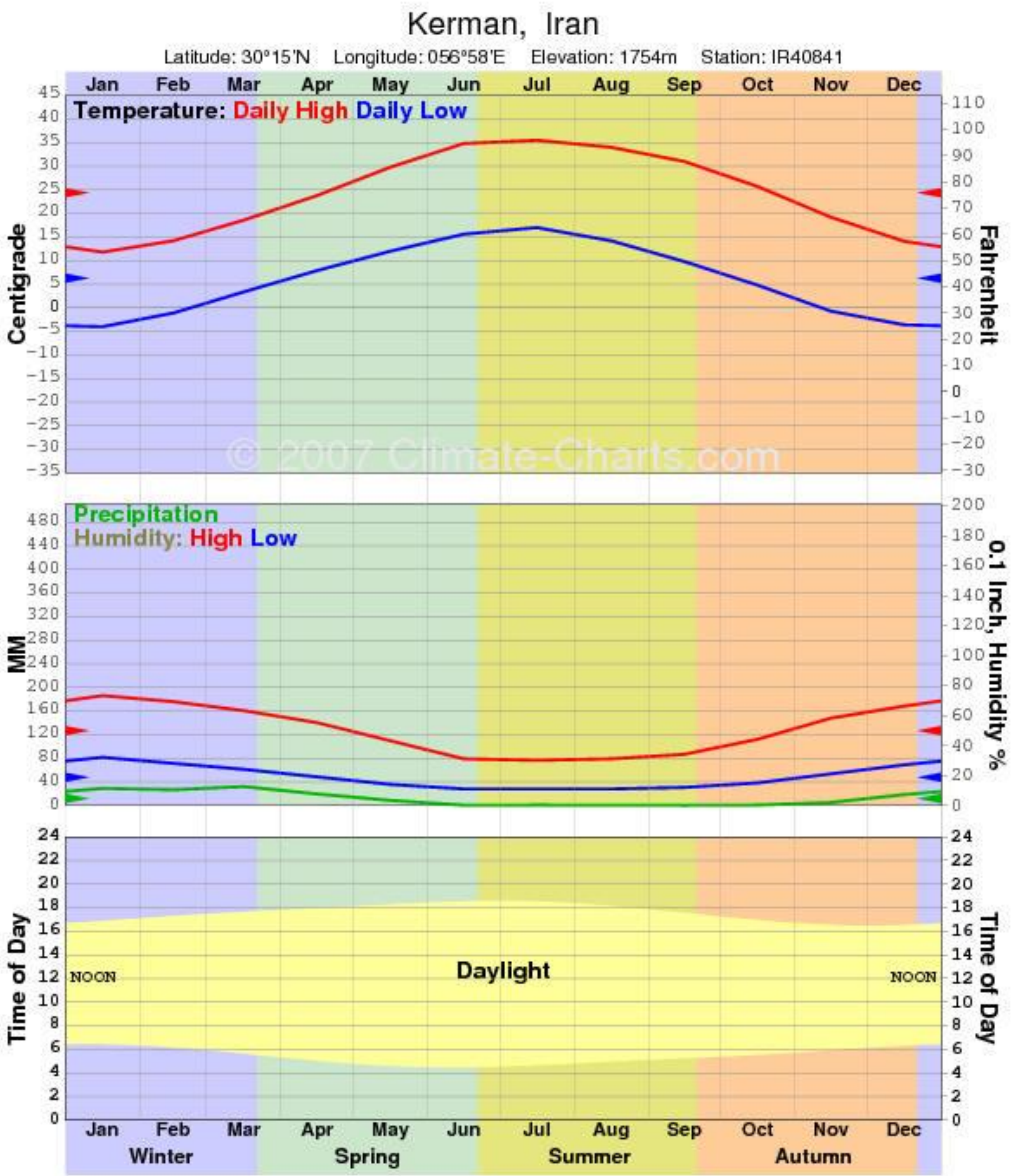


Figure 16 Kerman Province climate conditions and daylight hours

The Area 6 zone is approximately 1,750 m above sea level in an area of planar desert topography. The Area 6 ore body is generally of tabulate form, and about 5 km east of the presently mined Area 1. The closest distance between the two ore bodies is approximately 3.5 km. Figure 17 shows the location of the six anomalies in Golgohar area.

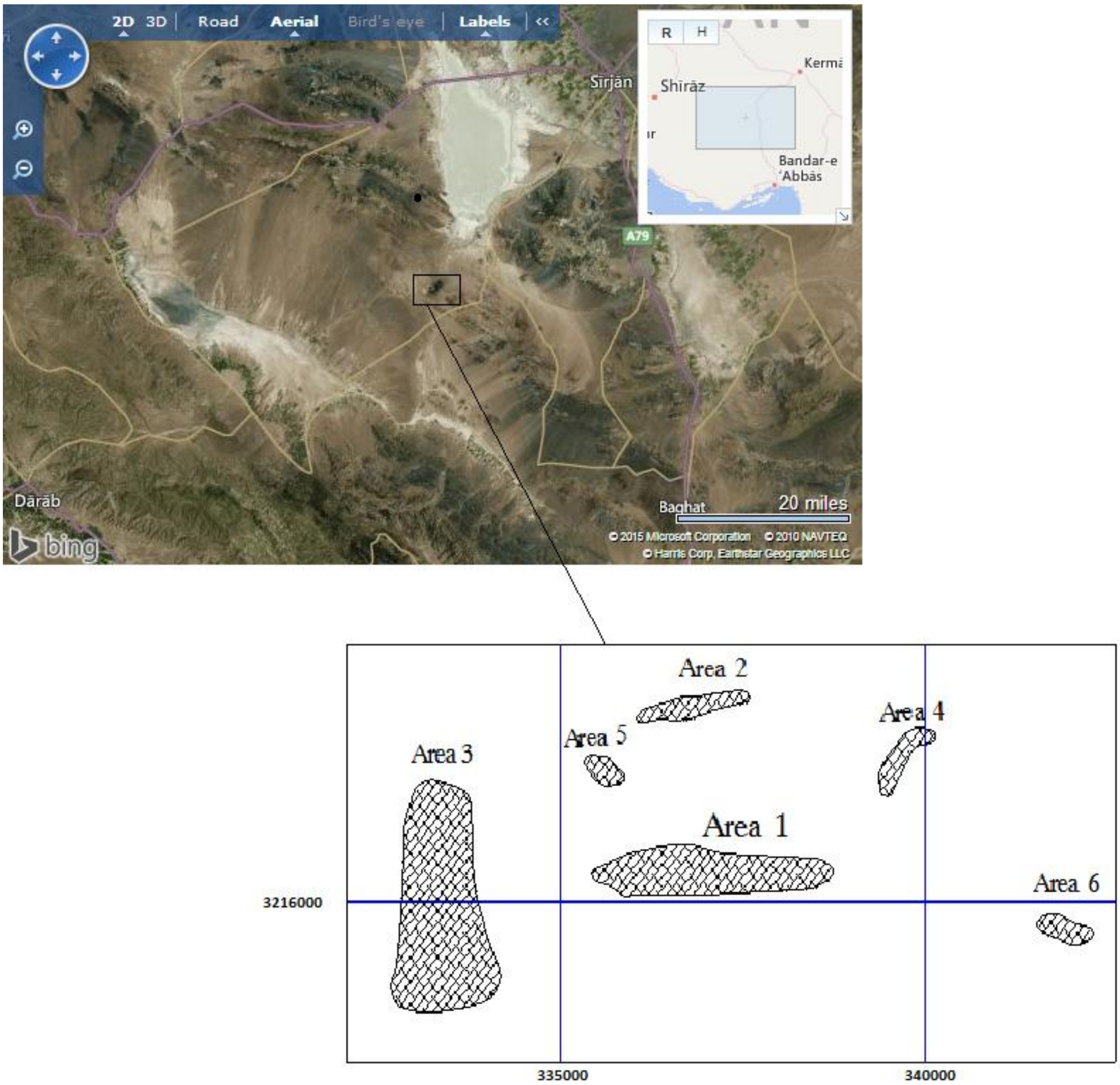


Figure 17 Location of six mines and anomalies in Golgohar

6.1.2 Geological Information

Babaki and Aftabi (1961) /15/ describes the geology of Golgohar as follows: “The Golgohar iron ore deposit is located in the eastern edge of the Sanandaj-Sirjan structural zone of Iran (Figure 18). The development of the 1,500 km long and 150 to 200 km wide Sanandaj-Sirjan zone was related to the generation of the Tethys Ocean and its subsequent destruction during Cretaceous and Tertiary convergence and continental collision between the Afro-Arabian and the Eurasian plates .The predominantly Mesozoic and Tertiary marine and continental sedimentary sequences and the major unconformities and the structural framework of the Sanandaj-Sirjan zone are comparable to those of the Central Iranian block in the east.

The host rocks of the ore deposit include metamorphosed sedimentary and volcanic rocks of the greenschist facies, probably of Upper Proterozoic-Lower Palaeozoic age. The most important host rock units include shale, sandstone, gabbroic-basaltic and diabasic sills, diamictite and cherty carbonatic sequences that have been changed to thick carbonate successions in the upper units. The structure of iron ore comprises macro-, meso- and microbanding of magnetite associated with shale, sandstone and cherty carbonates. The presence of diamictites and phenoclasts in magnetite banding and host rocks indicates an iron ore association similar to the Rapitan banded iron ore. Magnetite banding, granular, banded and massive textures all represent deposition of iron as hydromagnetite. The presence of organic matter (graphite) and microlayers of pyrite indicates variation due to reducing-oxidizing conditions controlled by oxygen amount at the time of formation of the iron ore. The sills of basic rocks in the region are of basalt characteristics that represent the process of inactive oceanic rifts at the time of ore formation. The upflow discharge of hydrothermal solutions into the seawater and sedimentary basin, followed by reaction with cold glacial water, causes hydromagnetite deposition within sediments and diamictites. The presence of massive magnetite texture, abundant tourmaline and low content of manganese indicates proximal ore mineralization in the central part of the volcanic-sedimentary activities formed.”

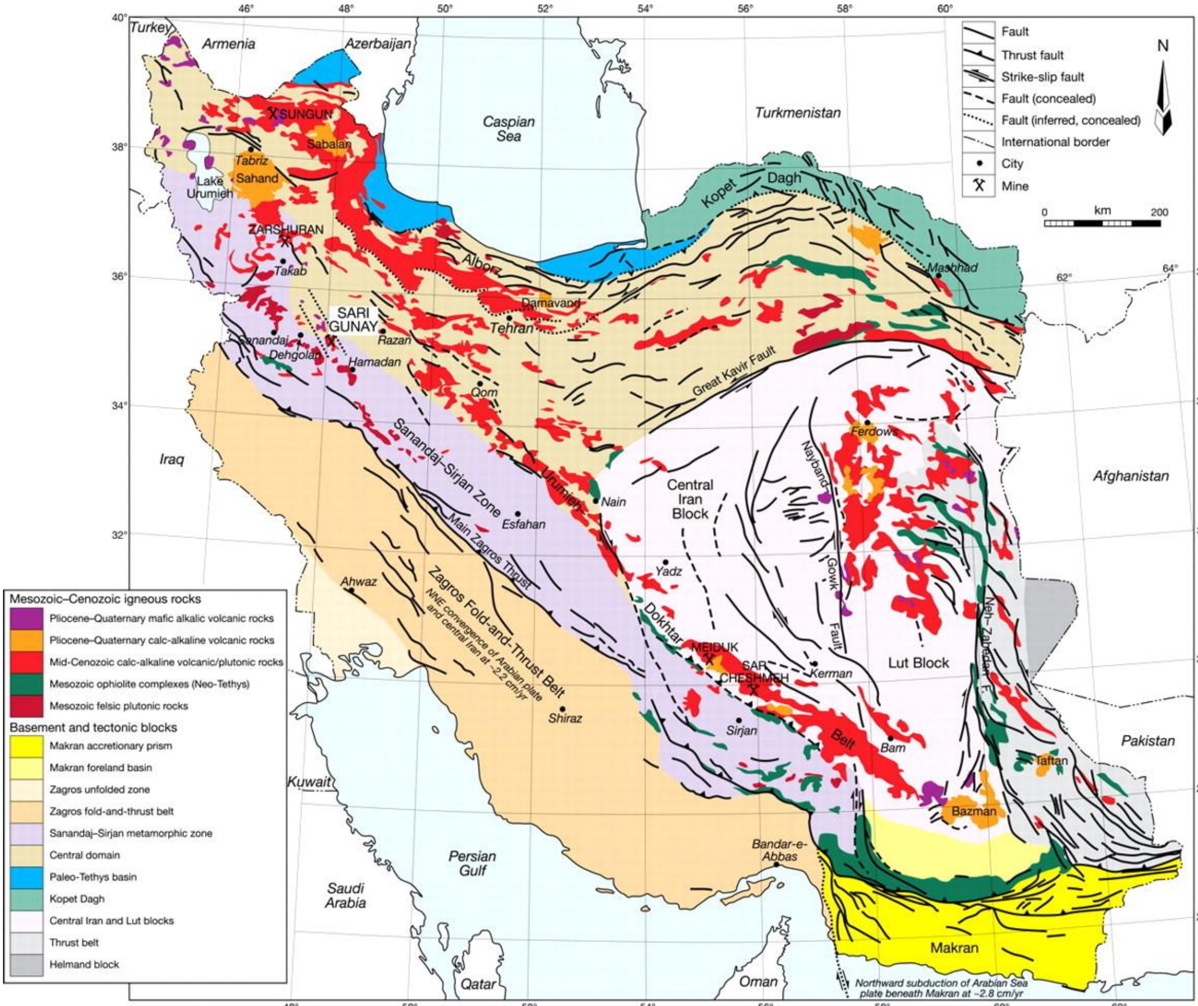


Figure 18 Sanandaj-Sirjan structural zone

"The major host rock for Golgohar iron ore deposit belongs to the eastern part of the Sanandaj-Sirjan magmatic-metamorphic zone. The complex consists of mica-schist, quartzite, marble, amphibole-schist, graphite-schist, calc-schist and amphibolite. Amphibolite forms intermittingly with the Golgohar orebody. Thus, determining the origin of this unit can shed some light on the origin of amphibolite and the conditions under which iron ore was formed. Field, petrographic and geochemical studies indicate that metapelites originated from iron-bearing shale while the amphibolites' protolith were marl-type sediments. In addition, the position of amphibolite samples on various diagrams (e.g. Mg vs. C, Cr, Ni and Si vs. Alk, Al, and Mg), which support their para-amphibolite origin, also confirms that the protolith of the Gol-Gohar amphibolites formed in a sedimentary environment" /15/.

The geological map of Golgohar 6 area (6.4 km²) in a scale of 1:5,000 and 8 geological sections were prepared by Kooshamadan Consulting Engineering Company.

6.2 Exploration

Resource evaluation is undertaken to quantify the grade and tonnage of a mineral occurrence. This is achieved primarily by drilling to sample the prospective horizon where the minerals of interest occur. The ultimate aim is to generate a density of drilling sufficient to satisfy the economy of an ore resource /16/.

The Iran Barite Company began to explore iron ore in Golgohar area in 1969. In 1974, the Golgohar area was transferred to the National Steel Company and exploration was carried out in cooperation with a Swedish company (Granges). The exploration activities started with airborne geophysics and continued with surface magnetometry and gravimetry in 74 km² in the Golgohar area. The result of these geophysical investigations indicated the presence of 1,100 million tonnes of iron ore in six anomalies in the Golgohar area and led to the opening and exploitation of Golgohar mine 1.

General exploration and the collection of increasingly detailed levels of deposit data in Anomaly 6 were restarted in 2005 by IMPASCO (IMIDRO Group) /17/. The general exploration activities in this step were:

- Geological map preparation for 6.4 km² (Scale 1:5,000)
- Topographical map preparation for 6.4 km² (Scale 1:5,000)
- Geological section drawings (8 sections)
- Exploration drilling (core drilling method) in 11 boreholes (11,792 m in total)
- Core cutting and sample taking and analysing the samples for Fe, FeO, P, S and SiO_2 (254 samples)

6.2.1 Airborne Surveying

Magnetic acquisition data using the airborne method was done by the Airoservice Company in 1975. The airborne surveying covered about 4,500 km² and resulted in the identification of six iron ore anomalies in the Golgohar area.

6.2.2 Magnetometry

Geophysical measurements (magnetometry) were carried out for about 640 acres in a network baseline of 20 m x 50 m (Table 14) by a proton magnetometer /17/.

Table 14 Specifications of magnetometry data acquisition

Total Points	Profile Spacing (m)	Point Spacing in Profile (m)	Area (km²)
7,275	50	20	6.4

After acquisition of the magnetic survey data, the database was completed and necessary corrections, processing and interpretation of the data was done and the following maps were prepared:

- Total intensity map. The maximum measured intensity is 47,300 nanotesla on N+200 to S-400 profiles. The intensity map is implemented by drawing 100 nanotesla frequency interval contour lines (Appendix 1). This shows that there is an anomaly in an east – west direction. The constituent elements of this source must be deep.
- Residual map. The natural base magnetic intensity of this area is 45,100 nanotesla. Figure 19 shows the residual intensity in Golgohar 6.
- Reduction to the pole. This correction caused the residual map and total intensity map to move to the north slightly (Appendix 1).
- Second derivative map. The second derivative map shows parts of the anomaly that are located in shallow or exposed areas, so this map does not

tracking any significant anomaly. According to the second derivation map of Golgohar 6 (Appendix 1), a small trace is in the East, which is too small and has no economic value.

- Upward continuation. The results confirmed the previous interpretation that the source of the anomaly is relatively large and deep. The upward continuation map also shows that the Golgohar 6 anomaly is at greater depth on the eastern side and has a relatively high density (Appendix 1).
- Modelling profile. Modelling determined that the depth of the Golgohar 6 anomaly is approximately 500 m and exploration drilling must be drilled deeper than 600 m.

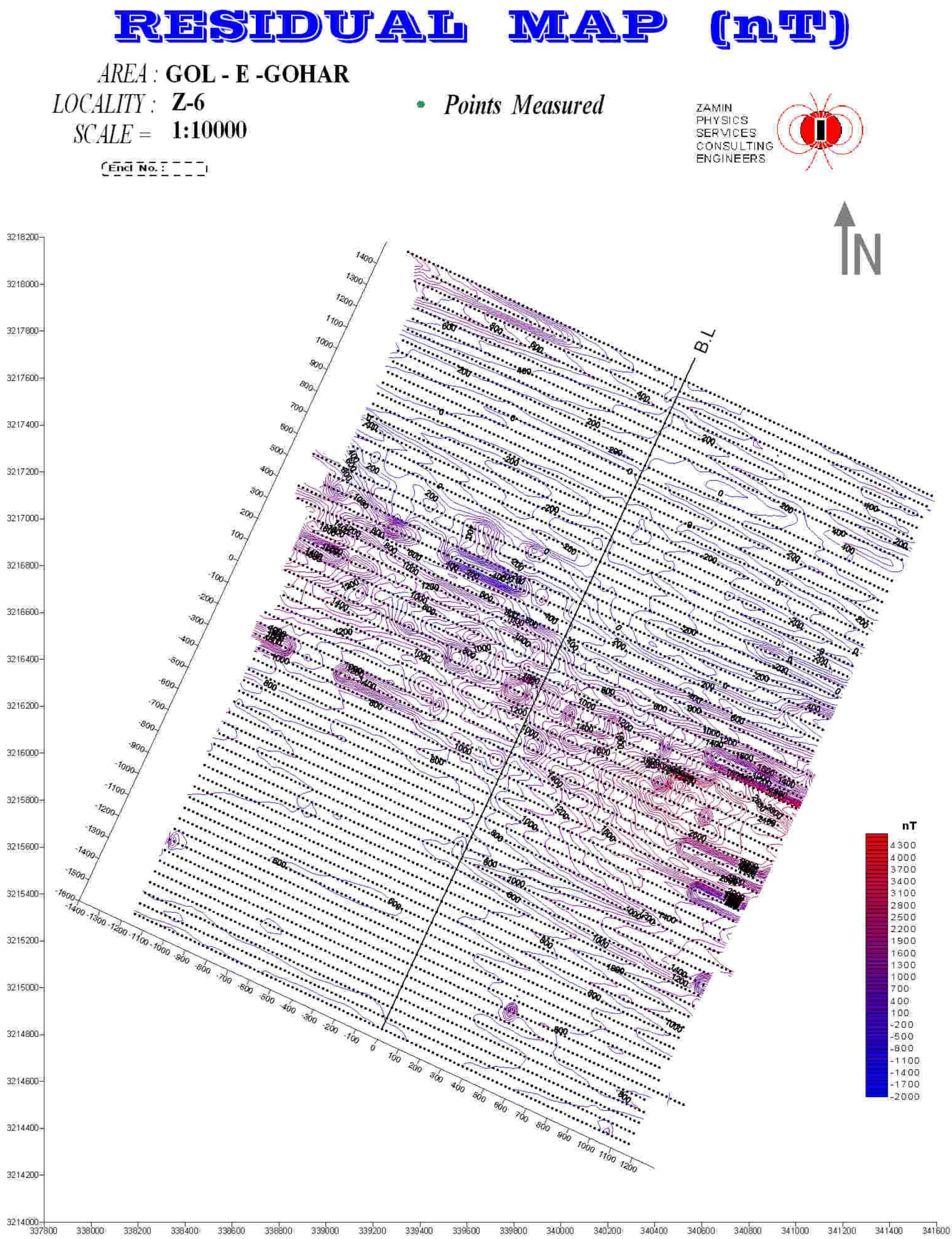


Figure 19 Residual magnetometry intensity map of Golgohar6

6.2.3 Core Drilling

The drilling method selection should be appropriate for the material being investigated, the objective of the programme and local drilling conditions. In Golgohar 6, the core drilling method was selected because core samples were needed for analysis, geotechnical investigation and a processing study. The drill hole size selected should provide sufficient representative sample material for analysis and reference. Drilling was done with HQ and NQ sizes in Golgohar 6 /17/.

A major objective is to obtain the most information and samples possible from each hole by optimizing location, drilling and sampling methods, depth, and completion.

According to the magnetometry results and interpretation of related maps, drilling point locations were determined (Figure 20). On the other hand, drilling points were located in a regular network (100 m x 100 m). Therefore, it was possible to draw longitudinal and latitudinal sections.

At this stage (general exploration of Golgohar 6), 18 boreholes were drilled. Table 16 shows a summary of the exploration drilling in Golgohar 6. Changes and additions to the exploration programme during or after exploration are expected, but can be minimized with careful planning, so hole depths and subsequent hole locations can be changed as exploration progresses. As an exploration programme evolves as data are acquired, information should be reviewed and added to maps and sections as the data become available. The total length of exploration drilling in Golgohar 6 is 11,792 m. Table 15 shows the length of boreholes and length of iron ore in each borehole.

Table 15 General specifications of Golgohar 6 exploration boreholes

Number of Boreholes	Total Drilling Length (m)	Total Drilling Length in Iron ore (m)	Iron Ore Thickness Average (m)	Overburden Average (m)
18	11,792	1,481	95	515

Drill holes should be logged (one part of core logging of Golgohar 6 is shown in Figure 21) and Core or sample recoveries should be noted on the logs as the holes are drilled.

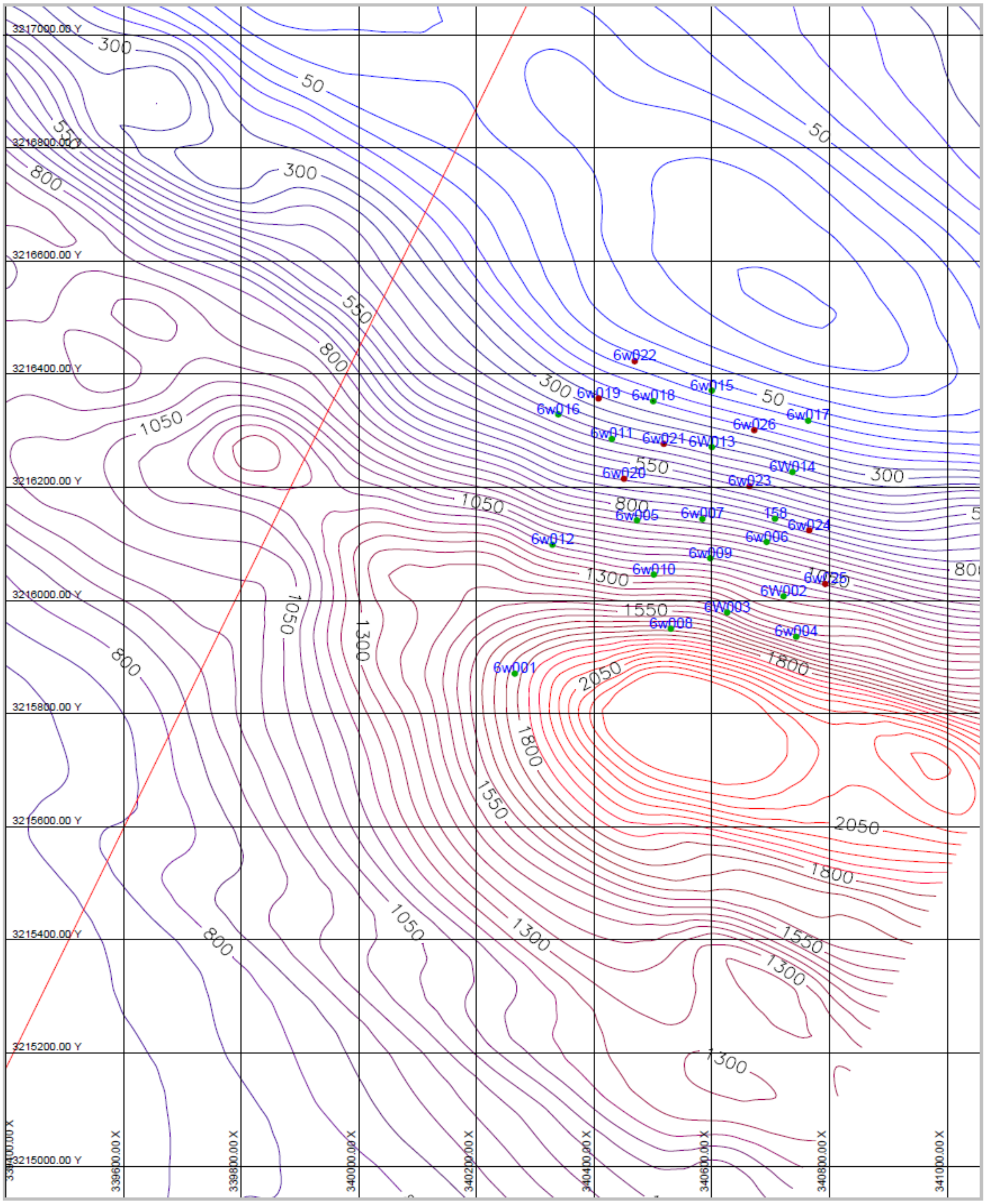


Figure 20 Drilling locations and residual magnetometry contour lines

Table 16 Golgohar 6 exploration boreholes

	Borehole No.	Borehole Length (m)	Iron Ore in Borehole (m)	Overburden (m)
1	6w001	651.3	0	0
2	6w002	636.3	126.5	497.5
3	6w003	625.35	118.55	477.8
4	6w004	658.8	113.2	526.5
5	6w005	545	26.1	489.2
6	6w006	679.04	128.05	522
7	6w007	636.56	108.3	507
8	6w008	594.78	33	464
9	6w009	619.94	109.58	486.55
10	6w010	544.1	35.3	507.8
11	6w011	700.21	132.4	528.4
12	6w012	634	0	0
13	6w013	723.35	149.8	526.9
14	6w014	707.78	89.3	547
15	6w015	675.67	56.3	545.6
16	6w016	685.88	60.7	580.9
17	6w017	731.51	0	0
18	6w018	749.08	193.8	527.7

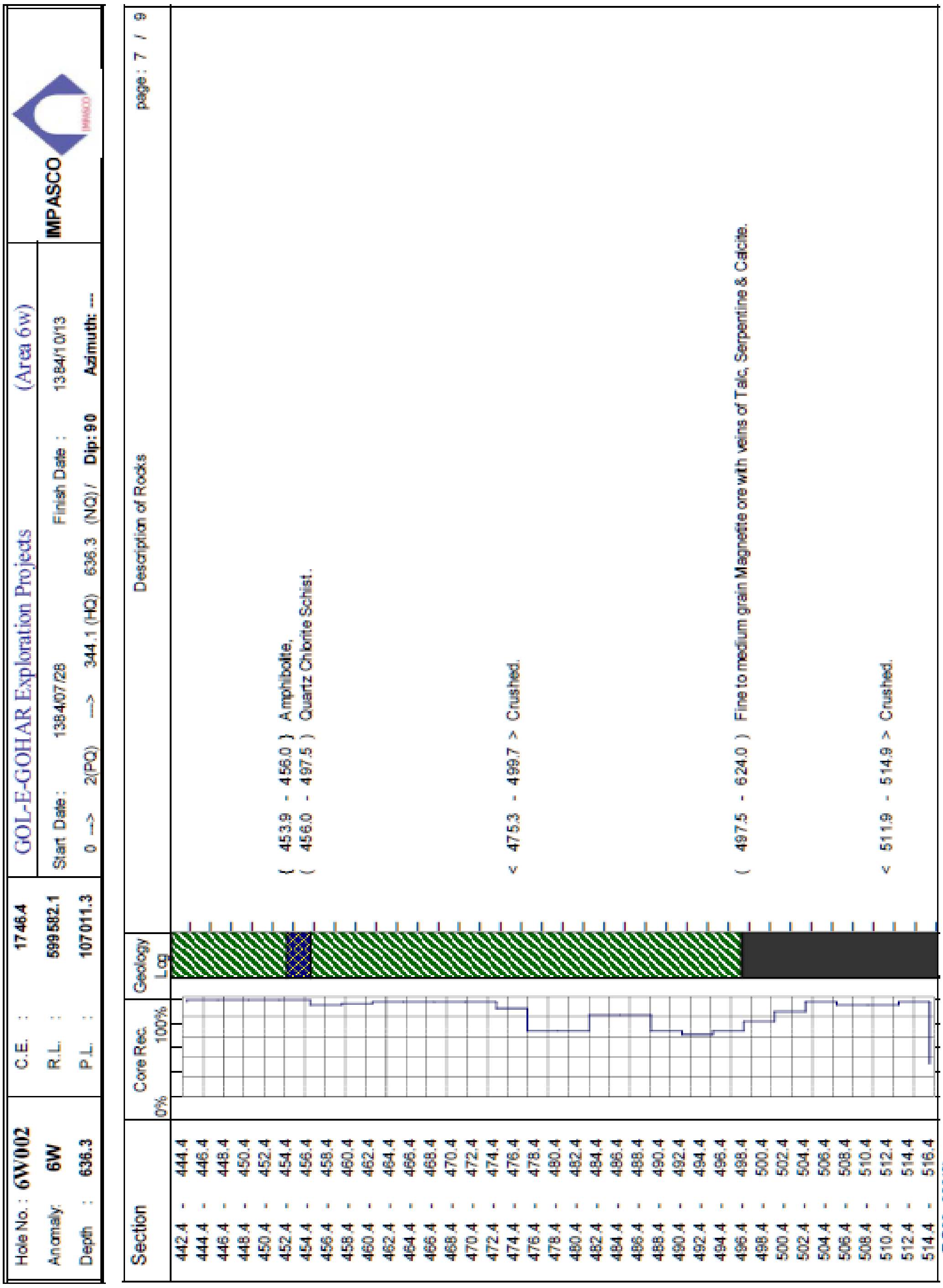


Figure 21 Core logging in borehole 6W002

6.2.4 Sampling and Analysing

Regular core sampling was done in Golgohar 6. First, all iron ore cores were cut into two halves using a core cutter machine. Then, a sample was taken from every six metres of core. In total, 254 samples were sent to the laboratory in order to determine the amount of Fe, FeO, P, S and SiO_2 /17/.

The result of the analyses is presented in Appendix 3. The average amount of elements in the Golgohar 6 iron ore is shown in Table 17 .

Table 17 Average percentage of selected elements in Golgohar 6

Fe (%)	FeO (%)	P (%)	S (%)	SiO_2 (%)
56.1	22.6	0.184	1.026	6.4

6.2.5 Resource Estimation

The aim of resource evaluation is to expand the known size of the deposit and mineralization and grade. A comprehensive and ongoing interpretation of all the exploration data is an essential activity at all stages of the project and should be undertaken to assess the results of the work. This interpretation should be based on all of the information collected to date, be systematic and thorough, describe and document the interpretation and discuss any information that appears at variance with the selected interpretation. The density of the exploration data should be critically assessed as to its ability to support the qualitative and quantitative conclusions/18/.

Estimation of a mineral resource and a mineral reserve are both fundamental steps in project development. The classification and categorization of these estimates could be done in accordance with the JORC Code (the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves produced by the Australasian Joint Ore Reserves Committee). A mineral resource can be estimated for material where the geological characteristics and the continuity are known or reasonably assumed and where there is the potential for production at a profit. Reserves can be estimated when a prefeasibility or feasibility study defined as economic and other relevant factors that indicate that these resources can be extracted. Resource estimation in Golgohar6 based on the parallel cross-section method is shown in Figure 22. Internal distances of these cross-sections are 100 m and their direction is east-west (Figure 23).

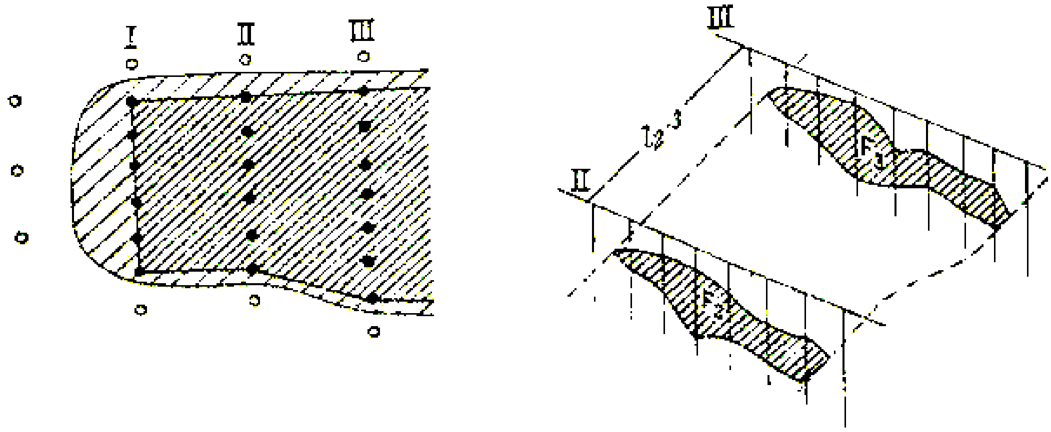


Figure 22 Parallel cross-section method

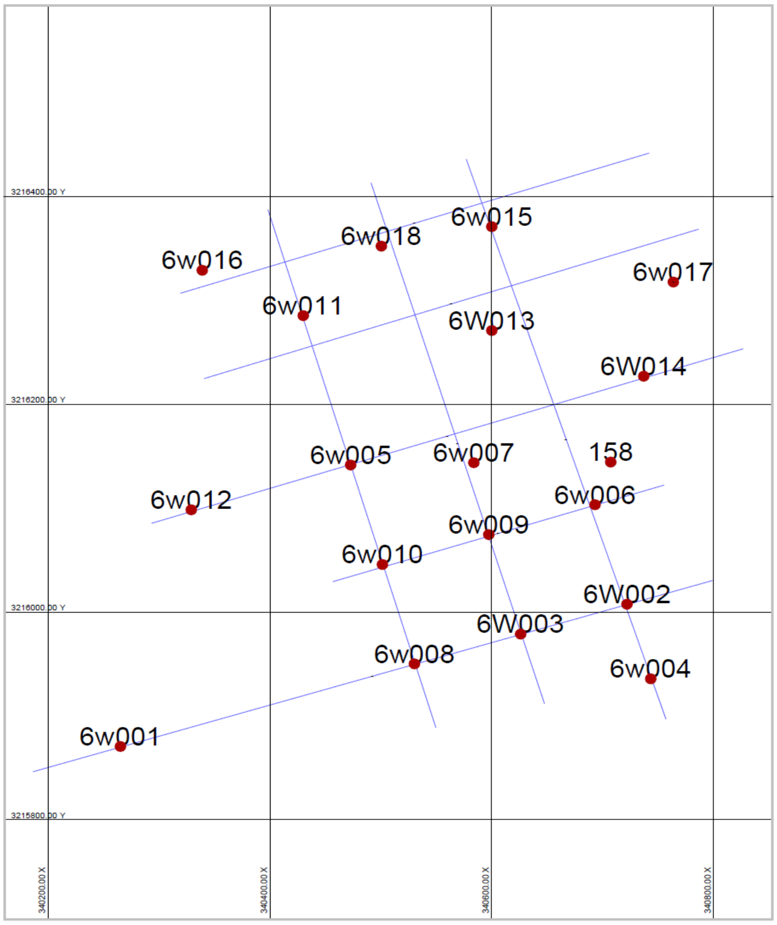


Figure 23 Overview of Golgohar 6 cross-sections

After providing digitized vertical and parallel cross-sections in Datamine software, a 3D model of the ore body was prepared by incorporating and correlating vertical cross-sections. Figure 24 shows the 3D ore body model of Golgohar 6.

The indicated resources have been estimated to be 65 million tonnes. It has been estimated that the Golgohar 6 ore body has a length of about 600 m (N-S) and an average width of nearly 400 m (E-W). The ore body thickness varies from 60 to 150 m, with an average thickness of 95 m.

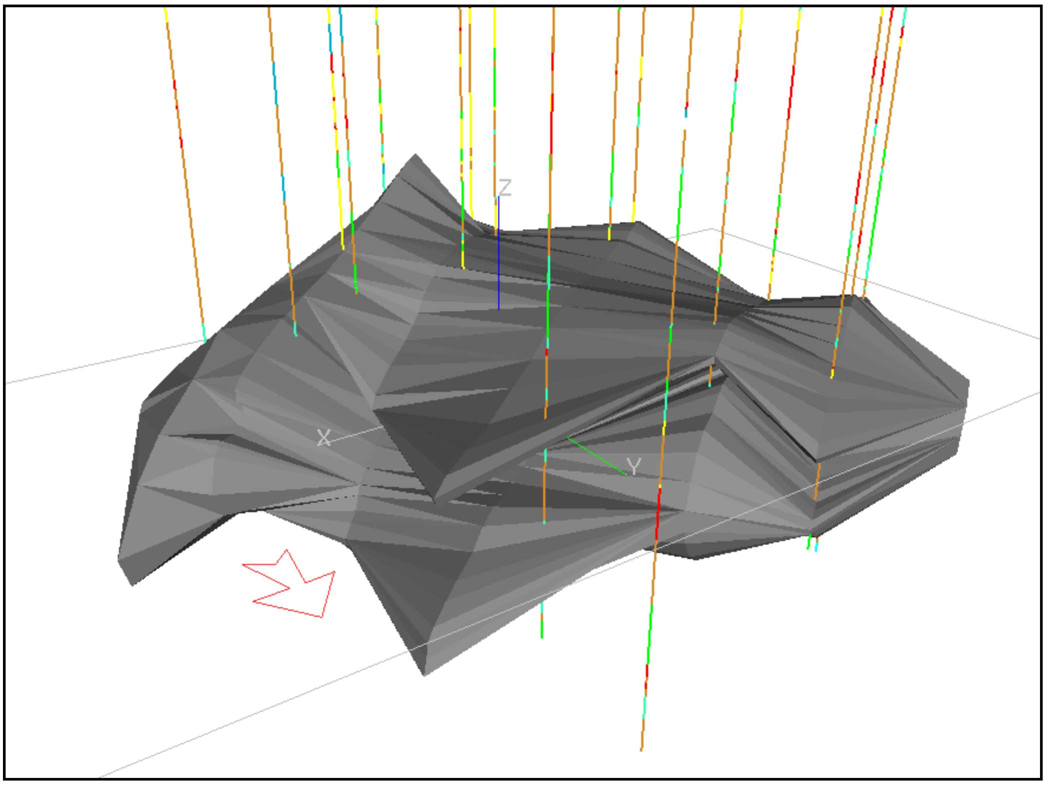


Figure 24 3D iron ore model of Golgohar 6

In Table 18, all available information on the ore deposit is summarized and the major factors affecting the selection of a mining method were determined based on this information.

Table 18 Deposit information

Parameters	Description
Deposit shape	Massive
Ore thickness	60 – 150 m
Ore dip	Horizontal
Depth	500 – 650 m
Grade	56.1% Fe
Reserve	65 million tonnes
RQD	66%

6.3 Extraction Method

The main aim of the present research is to choose the optimal extraction mining method. This section deals with the mining method selection for Golgohar 6 and is based on the qualitative method, numerical ranking method and fuzzy set theory.

The qualitative method has developed a flow chart selection process for defining the selection of a mining method (Figure 13) based on the geometry of the deposit (The deposit is tabular, flat very thick, and deep with uniform grade distribution) and the ground conditions of the ore zone. Based on this method, appropriate methods can be determined. These methods will be considered using the numerical ranking method (Nicholas method) and fuzzy set theory.

According to the numerical ranking method, the appropriate values corresponding to Golgohar 6 deposit are specified for each mining method (Table 19).

	General shape	Ore thickness	Ore dip	Grade distribution	Ore strength	Ore fracture spacing	Ore fracture strength	Hanging wall strength	Hanging wall fracture spacing	Hanging wall fracture strength	Footwall strength	Footwall fracture spacing	Footwall fracture strength	Sum
Open pit	3	4	4	3	4	4	4	4	3	4	4	3	4	48
Block caving	4	2	4	2	1	0	0	2	4	0	3	3	3	28
Sublevel stoping	2	4	4	3	4	4	4	3	0	4	2	0	4	38
Sublevel caving	3	4	4	2	3	4	2	2	4	0	2	1	4	35
Shrinkage stoping	2	4	4	2	4	4	0	2	4	0	3	3	3	35
Cut and fill	0	0	4	3	2	2	3	2	3	2	2	4	2	29
Top slicing	3	3	2	2	3	4	1	2	3	0	3	3	2	31
Square set	0	1	3	3	1	1	4	2	3	2	2	4	2	28

Table 19 Mining method selection for Golgohar 6 using the Nicholas method

The results of this table show that open pit mining and sublevel stoping are the most appropriate methods for the extraction of Golgohar 6.

The first step of decision making using the fuzzy theory is to build a set of possible alternatives or strategies in order to guarantee that the goal will be achieved (i.e., evaluating the alternatives). Based on qualitative investigation method, the possible methods are listed as alternatives in Table 20.

Table 20 Possible mining methods as alternatives

A1	Open pit
A2	Shrinkage
A3	Cut & fill
A4	Room & pillar
A5	Sublevel stoping
A6	Block caving
A7	Sublevel caving
A8	Longwall

The most important criteria relating to the mining method selection are listed in Table 21.

Table 21 Criteria for the selection of a mining method

P1	Ore strength	**P13**	Ore thickness
P2	Hanging wall strength	**P14**	Subsidence
P3	Foot wall strength	**P15**	Recovery
P4	Deposit shape	**P16**	Selectivity
P5	Deposit dip	**P17**	Dilution
P6	Deposit size	**P18**	Environmental risk
P7	Ore grade	**P19**	Production rate
P8	Ore uniformity	**P20**	Flexibility
P9	Depth	**P21**	Grade distribution
P10	Ore RMR	**P22**	Investment
P11	Hanging wall RMR	**P23**	Operating costs
P12	Foot wall RMR		

Twenty-three criteria are being dealt with at a given hierarchy and the procedure establishes a *23×23* pairwise comparison matrix, *P*, from which the relative weights of all the alternatives can be extracted (Table 22). Based on experience, engineering judgment and knowledge, a pairwise comparison of parameters was used to build up the matrix *P*. The equation below is a pairwise comparison matrix from which it can be seen that for *23* criteria, the pairwise comparison of element *i* to element *j* has one of the numerical values from Table 11 called a_{ij}. For consistency, $a_{ij} = k$ should automatically imply that $a_{ji} = 1/k$ /9/.

Table 22 Importance of each criterion matrix (Matrix P)

	P1	P2	P3	P4	P5	P6	P7	P8	P9	P10	P11	P12	P13	P14	P15	P16	P17	P18	P19	P20	P21	P22	P23
P1	1	0.2	5	1	0.33	0.33	1	0.2	0.14	0.33	0.2	1	0.11	3	1	5	0.33	0.33	0.2	3	1	0.33	0.33
P2	5	1	5	1	0.33	0.33	1	0.2	0.14	0.33	0.2	1	0.11	3	1	5	0.33	0.33	0.2	3	1	0.33	0.33
P3	0.2	0.2	1	0.33	0.2	0.2	0.33	0.14	0.11	0.2	0.14	0.33	0.11	1	0.33	3	0.2	0.2	0.14	1	0.33	0.2	0.2
P4	1	1	3	1	0.33	1	5	0.2	0.33	0.2	0.2	3	0.33	3	1	3	0.33	0.33	0.33	3	1	1	1
P5	3	3	5	3	1	3	3	1	0.33	1	1	5	1	5	3	9	1	1	1	7	3	1	1
P6	3	3	5	1	0.33	1	1	1	0.2	0.33	0.33	3	0.2	5	1	3	1	0.33	0.33	5	5	1	1
P7	1	1	3	0.2	0.33	1	1	1	0.2	0.33	0.33	1	0.33	1	0.33	1	0.33	0.2	0.14	1	0.33	0.2	0.2
P8	5	5	7	5	1	1	1	1	0.14	0.33	0.33	1	0.2	1	0.33	1	0.33	0.2	0.14	1	1	0.33	0.33
P9	7	7	9	3	3	5	5	7	1	3	3	7	5	9	3	7	5	3	3	7	7	3	3
P10	3	3	5	5	1	3	3	3	0.33	1	1	5	0.33	3	5	7	3	0.33	1	5	7	0.33	0.33
P11	5	5	7	5	1	3	3	3	0.33	1	1	7	0.33	3	5	7	3	0.33	1	3	7	0.33	0.33
P12	1	1	3	0.33	0.2	0.33	1	1	0.14	0.2	0.15	1	0.14	3	0.33	1	1	0.14	0.2	3	3	0.2	0.2
P13	9	9	9	3	1	5	3	5	0.2	3	3	7	1	7	5	7	3	1	3	7	9	1	1
P14	0.33	0.33	1	0.33	0.2	0.2	1	1	0.11	0.33	0.33	0.33	0.14	1	0.33	1	0.33	0.14	0.2	1	3	0.2	0.2
P15	1	1	3	1	0.33	1	3	3	0.33	0.2	0.2	3	0.2	3	1	3	0.33	0.2	0.33	3	5	0.33	1
P16	0.2	0.2	0.33	0.33	0.11	0.33	1	1	0.14	0.14	0.14	1	0.14	1	0.33	1	0.2	0.2	0.14	1	1	0.33	0.33
P17	3	3	5	3	1	1	3	3	0.2	0.33	0.33	1	0.33	3	3	5	1	0.33	0.33	3	5	0.33	1
P18	3	3	5	3	1	3	5	5	0.33	3	3	7	1	7	5	5	3	1	1	5	7	3	3
P19	5	5	7	3	1	3	7	7	0.33	1	1	5	0.33	5	3	7	3	1	1	3	3	1	1
P20	0.33	0.33	1	0.33	0.14	0.2	1	1	0.14	0.2	0.33	0.33	0.14	1	0.33	1	0.33	0.2	0.33	1	1	0.2	0.2
P21	1	1	3	1	0.33	0.2	3	1	0.14	0.14	0.14	0.33	0.11	0.33	0.2	1	0.2	0.14	0.33	1	1	0.33	0.33
P22	3	3	5	1	1	1	5	3	0.33	3	3	5	1	5	3	3	3	0.33	1	5	3	1	1
P23	3	3	5	1	1	1	5	3	0.33	3	3	5	1	5	1	3	1	0.33	1	5	3	1	1

The relative weights are determined from P by dividing the elements of each column by the sum of the elements of the same column. The geometric means of the i_{th} row, called M_i, is calculated as:

$$M_i=\prod_{J=1}^{n} a_{ij} \quad \text{for } i=1,2,3,....,n$$

The relative weights are calculated as the row average of the resulting normalized matrix. Table 23 shows the amount of *Mi, Wi* and *bi* .

$$W_i=(M_i)^{1/n}$$

$$Bi=\frac{\mathrm{Wi}}{\sum_{\mathrm{i}=1}^{\mathrm{n}} \mathrm{Wi}}$$

Through pairwise comparisons of the impact of the 23 alternatives on Matrix *P*, the values of M_i, W_i and eigenvector (*Bi*) would be calculated.
After that, the criteria weights are obtained in the eigenvector of the matrix.

Table 23 Values of Mi, Wi and Bi

M1	0.000003	W1	0.571656288790758	B1	0.018355216
M2	0.000065	W2	0.657528559862624	B2	0.021112475
M3	0.000000	W3	0.281989069753392	B3	0.00905434
M4	0.013254	W4	0.828633975465383	B4	0.02660647
M5	5735134.125000	W5	1.967197472447470	B5	0.063164415
M6	8.291750	W6	1.096328992651920	B6	0.035201844
M7	0.000000	W7	0.486103448000579	B7	0.015608214
M8	0.000976	W8	0.739784747840919	B8	0.023753625
M9	820620966493125.000000	W9	4.450789884578090	B9	0.142909668
M10	1371248.142476	W10	1.848542890719330	B10	0.059354555
M11	4479410.598754	W11	1.946174514645150	B11	0.062489392
M12	0.000000	W12	0.527669797978884	B12	0.016942861
M13	287097714225.000000	W13	3.148996781884900	B13	0.101110611
M14	0.000000	W14	0.387020335153604	B14	0.012426771
M15	0.071641	W15	0.891712067080364	B15	0.028631834
M16	0.000000	W16	0.337856877973946	B16	0.01084819
M17	223.653371	W17	1.265187350994280	B17	0.040623689
M18	35127696515.625000	W18	2.874112453215490	B18	0.092284396
M19	121307645.300625	W19	2.246318273097210	B19	0.072126658
M20	0.000000	W20	0.373054819092296	B20	0.011978355
M21	0.000000	W21	0.433845840499044	B21	0.013930284
M22	6820713.084375	W22	1.982080328076640	B22	0.063642286
M23	757857.009375	W23	1.801493483093810	B23	0.057843853

The importance and intensity of criteria in each extraction method are determined by experts and Matrix G (Table 24) is calculated based on the importance of parameters in each alternative [Appendix 4].

The membership grade of criteria (Matrix G) for alternative:

A1 are 0.504, 0.3071, 0.3071, ..., 0.0773, respectively

A2 are 0.3071, 0.1102, 0.143, ..., 0.2414, respectively

....

A8 are 0.0117, 0.04452, 0.2414 ,...., 0.1758, respectively

Table 24 Criteria intensity (Matrix G)

	A1	A2	A3	A4	A5	A6	A7	A8
P1	0.504	0.3071	0.1758	0.0445	0.504	0.4384	0.4384	0.0117
P2	0.3071	0.1102	0.2414	0.0445	0.1758	0.1758	0.3071	0.04452
P3	0.3071	0.143	0.143	0.2414	0.1102	0.0773	0.143	0.24144
P4	0.4712	0.1102	0.143	0.0773	0.3399	0.4384	0.3727	0.11016
P5	0.3727	0.2086	0.2414	0.1758	0.2743	0.3399	0.2086	0.07734
P6	0.3071	0.1758	0.1102	0.143	0.3727	0.4055	0.3727	0.1758
P7	0.4384	0.3071	0.2086	0.2414	0.4712	0.4712	0.4712	0.24144
P8	0,1758	0.1102	0.1102	0.0773	0.2414	0.143	0.1102	0.07734
P9	0.1102	0.1758	0.2086	0.1758	0.4055	0.4384	0.3727	0,24144
P10	0.3071	0.3071	0.2414	0.3399	0.3727	0.1758	0.3399	0.14298
P11	0.3399	0.2086	0.3727	0.1102	0.4055	0.4384	0.3399	0.30708
P12	0.1758	0.2086	0.2414	0.1758	0.2086	0.1758	0.143	0.1758
P13	0.2414	0.143	0.0773	0.0445	0.3727	0.3071	0.3071	0.04452
P14	0.4384	0.1758	0.3071	0.2414	0.1758	0.143	0.3727	0.14298
P15	0.2743	0.3071	0.3727	0.2086	0.3071	0.2414	0.2414	0.30708
P16	0.3727	0.1758	0.2414	0.3399	0.3071	0.2086	0.2743	0.1758
P17	0.3071	0.3071	0.4384	0.4055	0.2743	0.1758	0.1758	0.37272
P18	0,0445	0.2414	0.2743	0.2414	0.2743	0.2414	0.0773	0.1758
P19	0,1758	0.143	0.1102	0.2086	0.3399	0.3727	0.3071	0.24144
P20	0.3071	0.2414	0.2086	0.4055	0.3727	0.1758	0.3399	0.1758
P21	0.2414	0.2414	0.3399	0.2414	0.2743	0.2086	0.2414	0.11016
P22	0.1102	0.1758	0.143	0.2414	0.1758	0.1758	0.1102	0.14298
P23	0.0773	0.2414	0.2086	0.2086	0.2743	0.3071	0.2414	0.1758

The criteria weights (Matrix R) are calculated using the member of Matrix G and eigenvector B1, B2, B3, …., B23.

μD(Ai) = {(Gij)^B1, (Gij)^B2, (Gij)^B3,......., (G1j)^B23 } for j= 1 to 23 and i=1 to 8

The result of the criteria weights are shown in Table 25 (Matrix R) and the members are calculated by,

μD(A1) = {(0.504)^0.0182, (0.3071)^0.0211, (0.3071)^0.0.0091, ..., (0.0773)^0.0578}
μD(A2) = {(0.3071)^0.0182, (0.1102)^0.0211, (0.143)^0.0.0091, ..., (0.2414)^0.0578}
.....
.....
μD(A8) = {(0.0117)^0.0182, (0.04452)^0.0211, (0.2414)^0.0.0091,...., (0.1758)^0.0578}

Table 25 Weight of the parameters (Matrix R)

	A1	**A2**	**A3**	**A4**	**A5**	**A6**	**A7**	**A8**
P1	0.988	0.979	0.969	0.944	0.988	0.985	0.985	0.922
P2	0.975	0.954	0.970	0.936	0.964	0.964	0.975	0.936
P3	0.989	0.983	0.983	0.987	0.980	0.977	0.983	0.987
P4	0.980	0.943	0.950	0.934	0.972	0.978	0.974	0.943
P5	0.940	0.906	0.914	0.896	0.922	0.934	0.906	0.851
P6	0.959	0.941	0.925	0.934	0.966	0.969	0.966	0.941
P7	0.987	0.982	0.976	0.978	0.988	0.988	0.988	0.978
P8	0.960	0.949	0.949	0.941	0.967	0.955	0.949	0.941
P9	0.730	0.780	0.799	0.780	0.879	0.889	0.868	0.816
P10	0.932	0.932	0.919	0.938	0.943	0.902	0.938	0.891
P11	0.935	0.907	0.940	0.871	0.945	0.950	0.935	0.929
P12	0.971	0.974	0.976	0.971	0.974	0.971	0.968	0.971
P13	0.866	0.821	0.772	0.730	0.905	0.887	0.887	0.730
P14	0.990	0.979	0.985	0.982	0.979	0.976	0.988	0.976
P15	0.964	0.967	0.972	0.956	0.967	0.960	0.960	0.967
P16	0.989	0.981	0.985	0.988	0.987	0.983	0.986	0.981
P17	0.953	0.953	0.967	0.964	0.949	0.932	0.932	0.961
P18	0.750	0.877	0.887	0.877	0.887	0.877	0.790	0.852
P19	0.882	0.869	0.853	0.893	0.925	0.931	0.918	0.903
P20	0.986	0.983	0.981	0.989	0.988	0.979	0.987	0.979
P21	0.980	0.980	0.985	0.980	0.982	0.978	0.980	0.970
P22	0.869	0.895	0.884	0.914	0.895	0.895	0.869	0.884
P23	0.862	0.921	0.913	0.913	0.928	0.934	0.921	0.904

The final decision expressed in a membership decision function according to justification with the max-min composition method, can be determined as follows:

$\mu D(j) = \min \{(G1j)^{B1}, (G2j)^{B2}, (G3j)^{B3}, ..., (Gnj)^{Bn}\}$ for all j= 1 to m

Table 26 Minimum relative weights of criteria

	A1	A2	A3	A4	A5	A6	A7	A8
μD(j)	0.730	0.780	0.772	0.730	0.879	0.877	0.790	0.730

Therefore, the minimum relative weights of criteria for each alternative are listed in Table 26. The optimal solution, corresponding to the maximum membership 0.879, is A5, which selects the sublevel stoping method as preferable.

In the fuzzy decision-making section, fuzzy sets for each method (alternatives) and their factors have been suggested. To use the priority comparison method and select an appropriate mining method for this anomaly, the data of Table 25 (Matrix R) needs to be used. Based on matrix R and comparison elements by:

$$d_{ij} = Rij - Ri(j+k)$$

the matrix of comparison elements (Matrix T) is prepared (Table 27)

Table 27 Comparison elements for alternative A1 (Matrix T)

	A1-A2	A1-A3	A1-A4	A1-A5	A1-A6	A1-A7	A1-A8
P1	0,009	0,019	0,043	0,000	0,003	0,003	0,066
P2	0,021	0,005	0,039	0,011	0,011	0,000	0,039
P3	0,007	0,007	0,002	0,009	0,012	0,007	0,002
P4	0,037	0,031	0,046	0,008	0,002	0,006	0,037
P5	0,034	0,025	0,044	0,018	0,005	0,034	0,089
P6	0,019	0,034	0,025	-0,007	-0,009	-0,007	0,019
P7	0,005	0,011	0,009	-0,001	-0,001	-0,001	0,009
P8	0,011	0,011	0,019	-0,007	0,005	0,011	0,019
P9	-0,050	-0,070	-0,050	-0,149	-0,159	-0,139	-0,087
P10	0,000	0,013	-0,006	-0,011	0,030	-0,006	0,041
P11	0,028	-0,005	0,064	-0,010	-0,015	0,000	0,006
P12	-0,003	-0,005	0,000	-0,003	0,000	0,003	0,000
P13	0,045	0,094	0,136	-0,039	-0,021	-0,021	0,136
P14	0,011	0,004	0,007	0,011	0,014	0,002	0,014
P15	-0,003	-0,009	0,008	-0,003	0,004	0,004	-0,003
P16	0,008	0,005	0,001	0,002	0,006	0,003	0,008
P17	0,000	-0,014	-0,011	0,004	0,021	0,021	-0,008
P18	-0,127	-0,137	-0,127	-0,137	-0,127	-0,039	-0,101
P19	0,013	0,029	-0,011	-0,043	-0,049	-0,036	-0,020
P20	0,003	0,005	-0,003	-0,002	0,007	-0,001	0,007
P21	0,000	-0,005	0,000	-0,002	0,002	0,000	0,011
P22	-0,026	-0,015	-0,045	-0,026	-0,026	0,000	-0,015
P23	-0,059	-0,051	-0,051	-0,066	-0,072	-0,059	-0,042

For this purpose, an 8x8 matrix has been created (Matrix D) by first associating each method with a corresponding row and column of the matrix. The open pit mining method (A1) corresponds to Row 1 and Column 1, the shrinkage method (A2) to Row 2and Column 2, etc. The elements of $d_{11,}$ $d_{12,\ldots\ldots}d_{18}$ are the number of positive amounts in columns of Matrix T

$$d_{ij} = \sum_{i=1}^{m}\{[\mathrm{Rij} - \mathrm{Ri}(\mathrm{j}+\mathrm{k})\ ] \geq 0\}$$

and the elements of $d_{11}, d_{21,\dots\dots}d_{81}$ are the number of negative amounts of Matrix T. For filling of Matrix D, the comparison elements for alternatives A2,A3,…,A8 must be prepared.

$$D = \begin{vmatrix} d_{11}, d_{12,\dots\dots\dots\dots}d_{18} \\ d_{21}, d_{22,\dots\dots\dots\dots}d_{28} \\ \dots\dots\dots\dots\dots\dots\dots\dots \\ \\ d_{81}, d_{82,\dots\dots\dots}d_{88} \end{vmatrix}$$

The element d_{12} of the dominance matrix is the number of performance factors for which open pit mining is greater than the shrinkage mining method. When the set of paired ratings

{(0.988, 0.979), (0.975, 0.954), (0.989, 0.983), (0.77, 0.77), ...,(0.869, 0.895),(0.862, 0.921)}

is examined, it can be seen that open pit mining has a higher rating than the shrinkage method for 14 of the factors. Consequently, the value of *d*12 is 14. Similarly, d_{21} possesses a magnitude of 6. The complete dominance matrix is shown in Table 28. In this table, in the last row and column, cumulative values for each row and column have been calculated.

Table 28 Dominance matrix (Matrix D)

	A1	**A2**	**A3**	**A4**	**A5**	**A6**	**A7**	**A8**	**Total**
A1	0	14	14	13	7	13	10	15	86
A2	6	0	9	12	2	8	5	14	56
A3	9	12	0	14	7	11	8	17	78
A4	8	8	8	0	6	8	7	11	56
A5	15	17	15	17	0	13	17	20	114
A6	9	13	12	13	7	0	11	17	82
A7	9	13	13	14	4	7	0	17	77
A8	7	5	5	6	2	3	6	0	34
Total	63	82	76	89	35	63	64	111	

The difference of each alternatives in rows and column in Table 28 are 23, -26, 2, -33, 79, 19, 13 and -77 for OP, SH, CF, RP, SS, BC,SC and LW . Therefore based on justification using priority comparison the alternative SS (sublevel stoping method) and OP (open pit mining) are the selected methods.

6.4 Preferable Methods

The mining method selection (the numerical ranking method and decision-making method by max-min composition justification and decision-making method by priority comparison justification) is summarized in Table 29. A comparison of the results shows that the preferable methods are underground mining using the sublevel stoping method and the open pit mining method.

Table 29 Preferable extraction method

Mining Method Selection	First Priority	Second Priority
Numerical ranking method	Open pit mining	Sublevel stoping
Justification using the Max-min compositon method	Sublevel stoping	
Justification using priority comparison	Sublevel stoping	Open pit mining

6.5 Open Pit Mining Assessment

Open pit mining involves the basic procedures of overburden removal, drilling and blasting, ore and waste loading, hauling and dumping and various other auxiliary operations.

According to the investigation of various methods, one of the preferable methods is open pit mining in Golgohar 6. Also, all iron ore mines in Iran and almost all iron ore in the world are extracted using the open pit mining method. Therefore, the possibility of extraction using the open pit mining method (by shovel & truck and by semi crusher & conveyor) should be studied for Golgohar 6.

For this purpose, the following parameters have been considered:

- Due to the large-scale mining extraction, as well as compliance of bench height with drilling and loading machinery and appropriate efficiency, the bench height and slope of the bench is considered 20 m and 75°, respectively.
- Although the overall slope angle of the mine must be calculated based on the results of the study of joints, geomechanical tests and hydrogeological assessment, for this preliminary purpose, the overall mining slope angle is considered 58° based on experience with Golgohar 1, 2, 3 & 4.
- According to the overall slope angle and bench slope, the berm width will be 12 m.
- The volume of iron was calculated in Section 6.2.5 (15 million m^3) and based on primary modelling (Figure 25), the total volume of iron ore and overburden (gangue) will be 440 million m^3.
- The stripping ratio based on the density of iron ore and overburden (alluvium, schist, quartz and gneiss), which are 4.3 and 2.5 tonnes per m^3 respectively, is 1:16.
- For the open pit mining method, the first pit stage could be considered down to 1,300 m elevation with a bottom pit radius of 50 m. The volume of the first pit was calculated using Datamine software. This volume is 77 million m^3. If the mine life time is 20 years, the first pit (waste material) will be extracted for three years. After that, the extraction of iron ore will start and extraction of waste will be continued.
- Drilling and blasting activities are needed in both systems (dump truck and shovel haulage system & in-pit crushing and conveyor haulage system).
- Annual extraction of iron ore and/or waste rock is 56.3 million tonnes.
- Distance of the pit centre to the dump is 3,400 m.

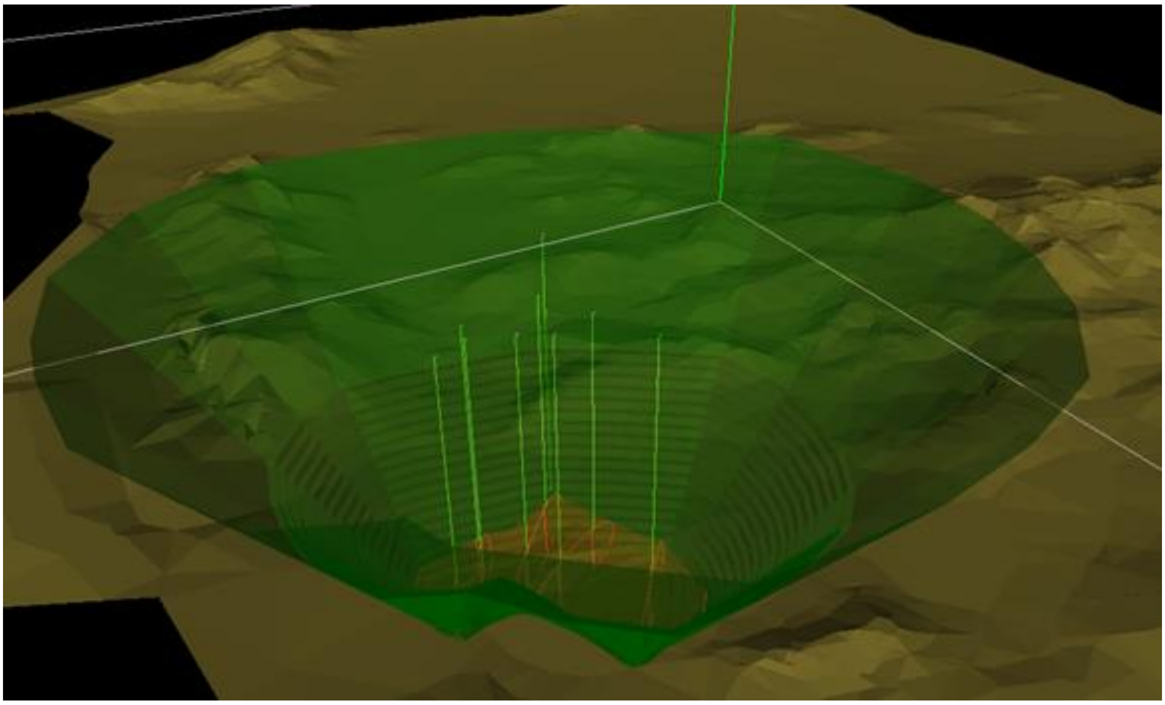


Figure 25 Primary modelling of Golgohar 6 open pit mine

The operation tasks in open pit mines consist of drilling, blasting, loading, haulage and general services. Among them, the haulage is the most expensive operation, accounting for more than 50% of the total operation cost in open-pit mines. Therefore, minimizing the haulage cost can be one of the most critical constraints for ore production. Two haulage systems, Dump truck & Shovel system and In-pit Crushing & Conveyor haulage system are considered.

6.5.1 Dump Truck & Shovel Haulage System

The loading of ore and waste is carried out simultaneously at several different locations in the pit. Shovels of various sizes are used to load material onto trucks. Hauling material from the shovel production faces to the dumping sites must be accomplished through a network of haul roads of various length and gradients. Haul roads can be extremely complex, cover large surface areas and pass through extreme elevation changes. The loading times of shovels depend on shovel capacity, digging conditions and the truck capacity. The number and type of trucks and shovels are two important factors in determining the optimum design parameters of an open pit mining system. In this case, the specifications are as below:

- Dump truck: Cat 785c. Figure 26 shows the specifications of this truck
- The dump truck haulage cycle time was estimated using XTracktor software (Figure 26 and Figure 27)
- Shovel loader (excavator): Liebherr R994. Loading capacity 15 m^3 (26.7 tonnes)

Load and haulage machinery specifications and compatibility are shown in Table 30.

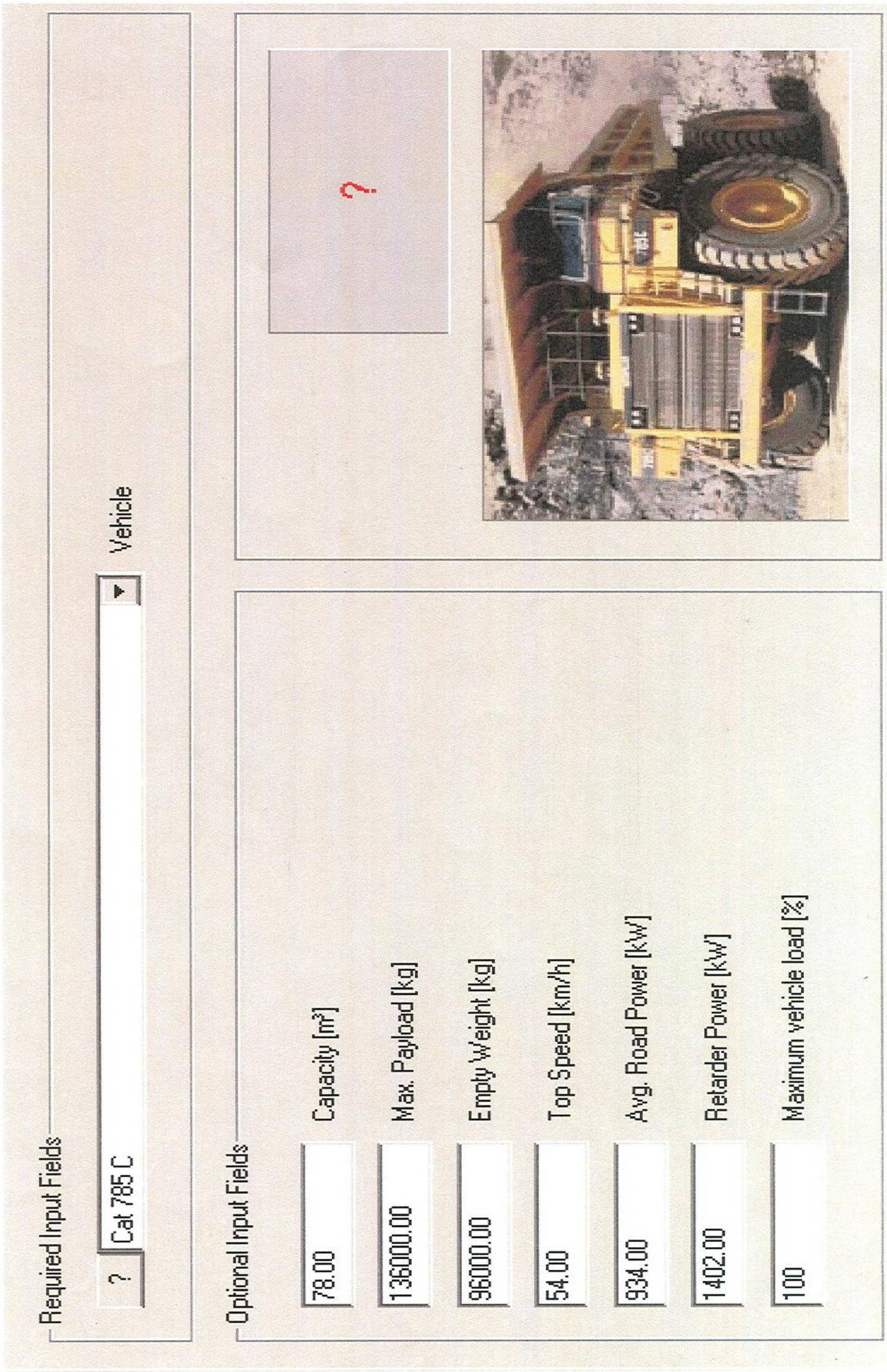


Figure 26 Specifications of the recommended dump truck

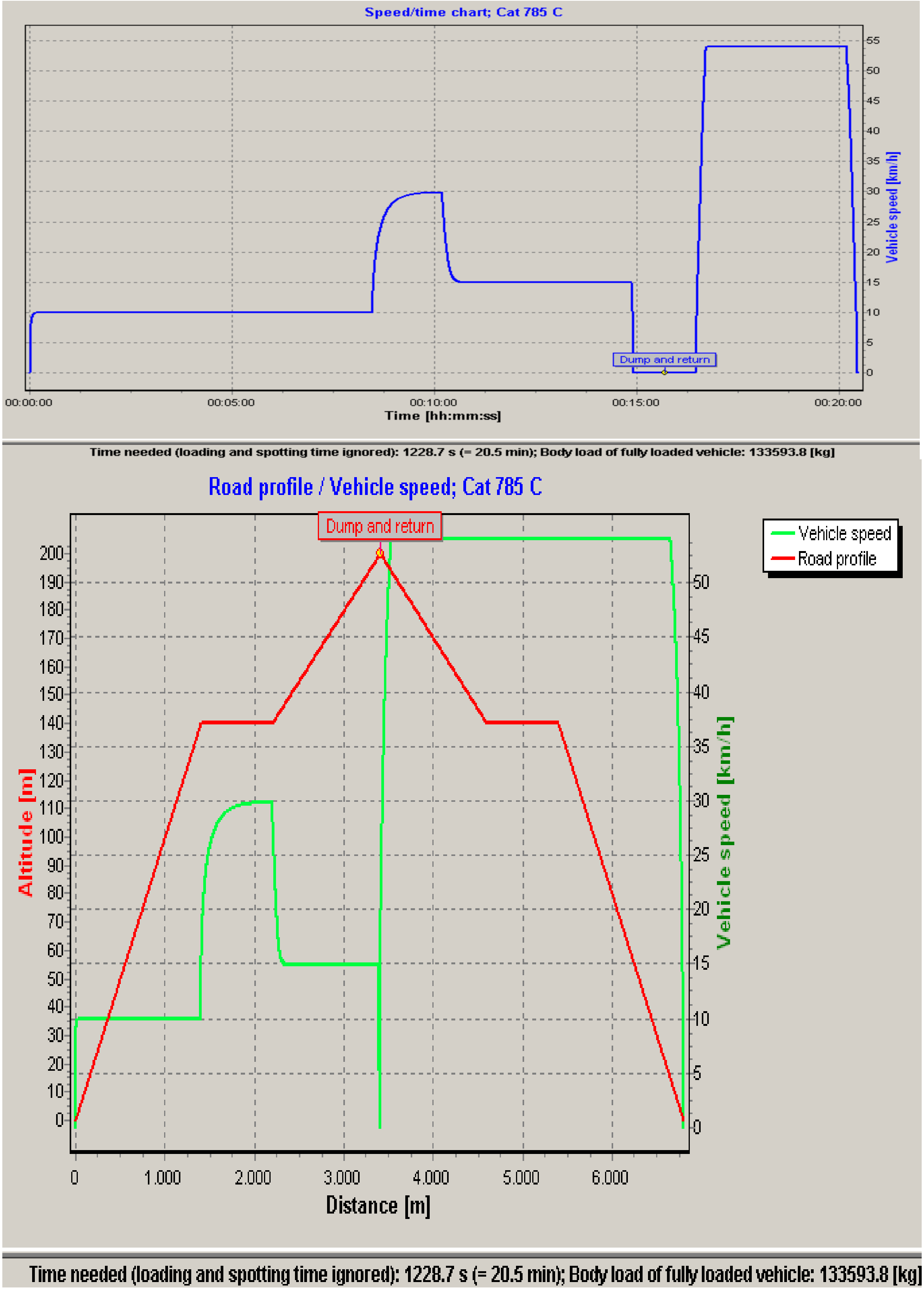


Figure 27 Dump truck haulage cycle time

Table 30 Specification of excavator and load & haul cycle

Vehicle payload:	136000 [kg]
Maximum vehicle load:	136000 [kg] (100 [%])
Excavator bucket size:	15 [m³]
Bucket load:	26700 [kg]
Number of loading passes:	5
Calculated vehicle load:	134000 [kg]
Vehicle body:	78 [m³]
Density of loaded material (loose):	1.88 [t/m³]
Volume of loaded material:	71 [m³]
Working time per day	16 [h]
Working time per year	4000 [h]
Vehicle loading time:	103 [s] = 00:01:42 [h]
Vehicle spotting time:	45.0 [s] = 00:00:45 [h]
Sum of transport time, loading time, spotting time:	1376.2 [s] = 00:22:56 [h]
Minutes per hour:	50 [min/h]
No. of vehicles needed for maximum production:	10
Maximum production per hour:	2717160 [kg]
No. of total vehicles needed	50
No. of total excavators needed	5

6.5.2 In-Pit Crushing & Conveyor Haulage System

The rising operating costs and declining commodity prices at most properties have forced the owners to look at various alternatives to cut costs in order to stay competitive. Haulage costs have risen significantly with the increase of diesel prices. One alternative to reduce haulage costs is to shorten the truck haul distance by bringing the truck dump point into the pit. Using an in-pit movable crusher or crushers and conveying the ore and/or waste out of the pit can reduce the haul costs.

The potential effects of installing in-pit crushing and conveyor haulage on open pit operations and on mine planning requirements can be significant. This will effect pit geometry, operating strip ratio, and mine access requirements when movable crushers and conveyors are brought into the pit. Details of mechanical installations including belt widths, tensions, and crusher sizes will not be discussed because this equipment will vary with production requirements and many technical papers have recently discussed mechanical installations.

As a general rule, conveyor installations work especially well when large volumes of material have to be moved from one single source to a single destination.

The flow of material utilizing an in-pit movable crusher and conveyor system starts with the trucked material being dumped into the feeder pocket. The material is crushed and fed onto horizontal transfer belts in the pit or directly onto a major upslope belt taking the material out of the pit. There may be more than one of either belt type with transfers at each belt junction, depending on the pit geometry and depth.

The first step when evaluating the potential installation of movable conveyors and crushers is to establish the geometric requirements of the installation being considered.

Movable crushers and feeder installations are offered by a wide range of manufacturers. The ideal situation is to establish the necessary access geometries (haul roads, crusher sites, beltways etc.) while mining at the planned rate.

In-pit crusher locations are generally selected with the primary goal in mind being to shorten the truck haul profiles. Shorter hauls in the pit can reduce the size and number of the required truck fleet, thus lowering truck capital and replacement capital costs. The calculation of optimal central locations for the crusher to minimize haul distances is necessary /18/. The optimal location for an in-pit crusher is where it will be out of the way of mining for the longest period of time. The corners and flat areas of a pit that do not change over a long period of time can be utilized as crusher locations.

The specifications of an in-pit crusher and conveyor system in Golgohar 6 are listed below:

- For loading and hauling of blasted iron ore and waste rock, five shovel excavators and 15 dump trucks are needed.
- Two semi-mobile crushers (gyratory crusher) with each with a capacity of 7,000 t/h.
- Crushers' discharge size: 250 mm.
- During the mine lifetime, the crushers are moved a total of four to five times.

Conveyor specifications are listed in Table 31 .

Table 31 Specifications of the belt conveyor

1	Conveyor width	1,800 mm
2	Conveyor length	3,000 m
3	Maximum longitudinal inclination	25%
4	Trough angle	30°
5	Surcharge angle	15°
6	Belt clear edge distance	122 mm
7	Capacity factor	0.86
8	Belt speed	4 m/s
9	Conveying capacity	14,000 t/h

6.6 Sublevel Stoping Method

Sublevel stoping (also known as blasthole, longhole, open, or vertical crater retreat (VCR) stoping) is classified as a large-scale and unsupported method. Sublevel stoping is very development intensive, although the cost of development is compensated by the fact that much of it is done in ore.

The sublevel stoping method is especially suitable for ore bodies with the following characteristics:

- The rock in the ore bodies and in the host ground is reasonably competent.

- The ore zones have relatively large horizontal and vertical dimensions.
- The number and size of barren or waste zones within the ore body is minimal.

Sublevel stoping is a mining method in which ore is blasted from different levels of elevation but is removed from one level at the bottom of the mine /20/. Before mining begins, an ore pass is usually drilled from a lower to a higher elevation. The bottom of the stope is V-shaped to funnel the blasted material into the drawpoints.

After blasting, the ore falls down to the lower drift where scoop trams or LHDs can drive in to load and dump it at an ore pass. Drilling and blasting continues until the stope is completely excavated (Figure 28). Sublevels are prepared and connected via an inclined ramp. The sublevels are worked simultaneously, the lowest on a given block being the farthest advanced and the sublevels above following one another at short intervals.

Once the stope is completely extracted, it is backfilled from the bottom up. The back-fill material used can be a mixture of sand and rocks, waste rock with cement, or de-watered mill tailings (rejected low grade ore from processing, usually fine and sandy). The backfill material must have plenty of strength to support the roof of the empty stope.

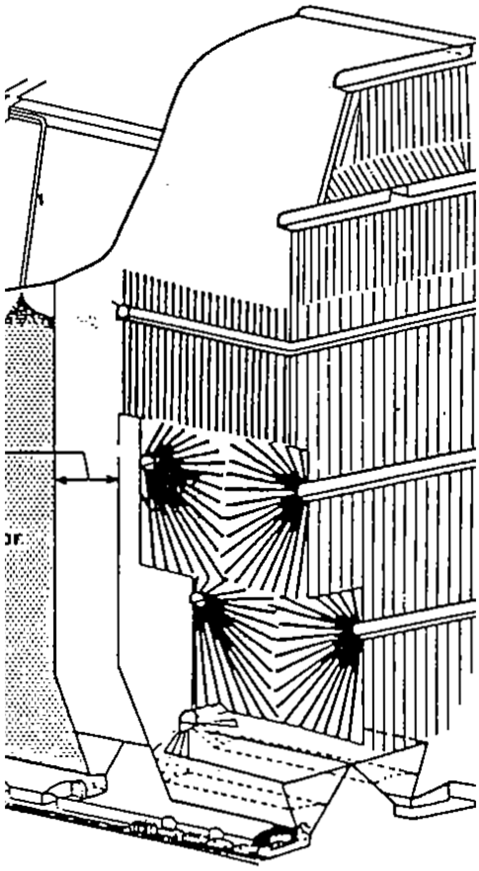


Figure 28 Sublevel stoping method

The drill rig (Figure 29) continues drilling until all the rings on one level are completed. It is then transferred to the next sublevel to continue drilling.

The blasted ore drops to the stope bottom to be recovered by the LHD vehicles mucking in the draw-point beneath the stope.

Normally, the long-hole drilling stays ahead of the charging and blasting, providing a reserve of ready-to-blast ore, thus making for an efficient production schedule.

In Golgohar 6, based on the ore body shape, sublevel stoping was considered, with stopes 40 m wide and 80 or 100 m high on average. Precise stope dimensions depend on local conditions. The ore extraction system from the stope consists of a series of drawpoints on 10 m centres driven from the haulage to crosscuts. Sublevels are driven on 20 m vertical intervals and are tied to the footwall ramp system.

Figure 29 Drilling rig on a sublevel

The sublevels are designed primarily to give access for longhole drilling in the stopes, but have also been used for production to facilitate draw control. The ring shape stoping holes are drilled using rotary drills, drilling 51 mm diameter holes according to the pattern shown in Figure 30. The specifications of drilling and blasting at the sublevels are presented in Table 32 .

Table 32 Drilling and blasting specifications

Parameter	**Description**	**Unit**
Hole diameter	51	mm
Drilling type	Ring drilling	
Production drift cross section	3×3	m
Vertical distance between sublevels	20	m
Horizontal distance between production drifts	20	m
Hole length	8 – 12	m
Spacing in front holes	Min:0.1, often 0.5	m
Spacing in end holes	Max 2.5	m
Overburden	1.5	m
ANFO consumption	1.9	kg/m of hole
Primer consumption	0.14	cartridge/m of hole
Cordtex consumption	1.5	m/m of hole

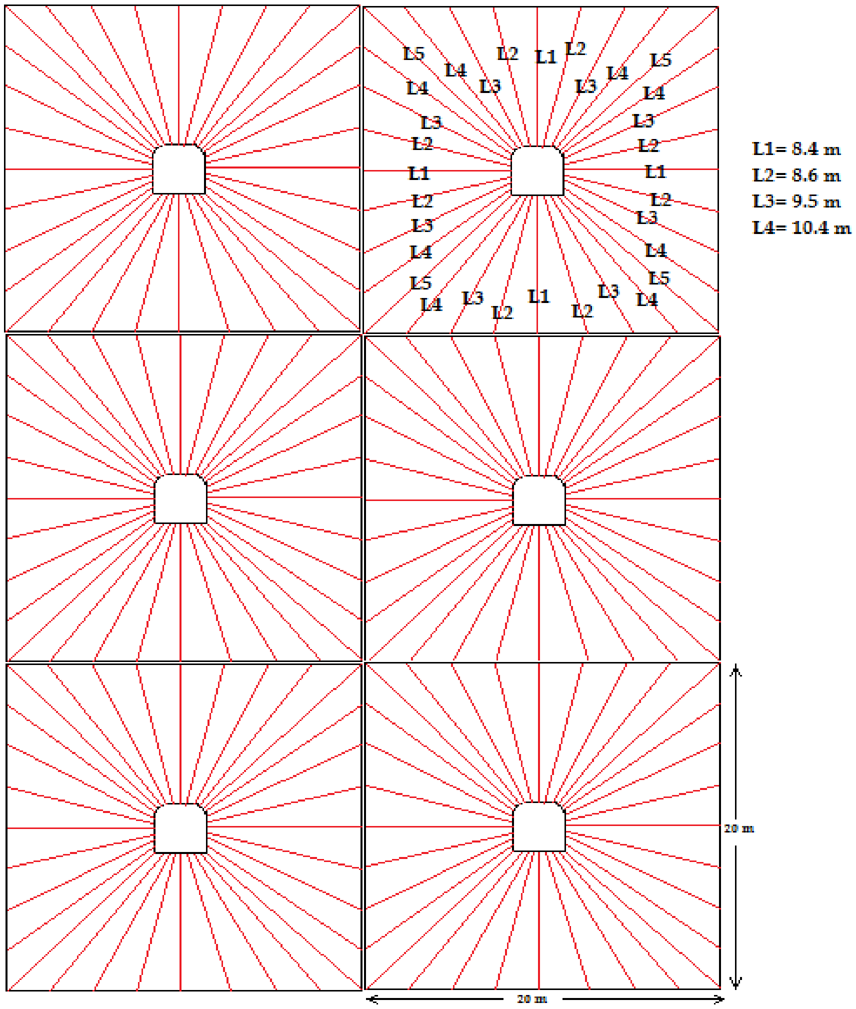


Figure 30 Drilling pattern in sublevels

6.6.1 Mine Access and Development

It is very important for a mine to have a reliable access to the deposit. The type and location of the portal significantly affects mine operation. Transportation costs are lower if ore is hauled down gradient and drainage is easier when the final destination is lower than the origin. Transportation, both inside and outside the mine, is affected by portals. Conveyor belts, loading chutes and other elements of ore hauling network inside the mine are planned on the basis of portal location and size.

Mine development normally starts from a shaft or slope in the footwall to avoid any subsequent caving effects from the stopes.

In the selection of an option for mine access (declines vs. shafts) in Golgohar 6, shafts have certain advantages over declines:

- Even at the maximum angles for ore transport by conveyor belt, declines are about three times in length for the same depth, compared to shafts.
- The capital cost of declines is higher than that of shafts (Figure 31).
- If the strata are poor, the increased length results in higher maintenance costs.
- Increased length causes greater pressure drops affecting the ventilation.

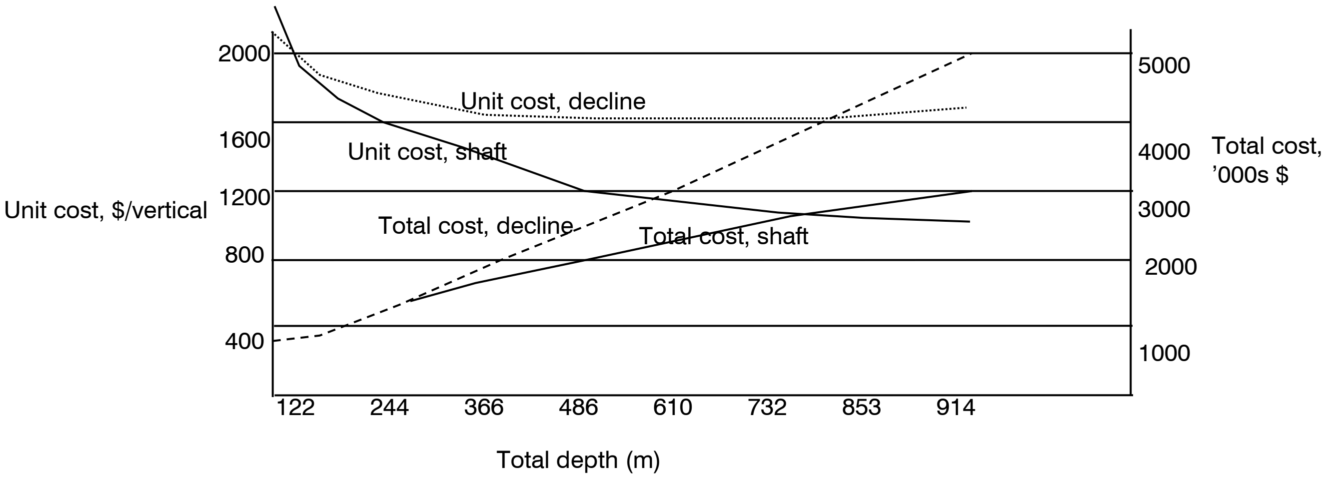


Figure 31 Comparison of costs for a decline and for a shaft /33/

6.6.2 Transport and Service Shafts

Shafts are the most important item of capital in deep mines, providing all services for underground operations, including fresh air, transportation of ore and supplies, personnel traffic, power (electricity and compressed air), communications, backfilling, and dewatering [65].

A central location of shafts in the mining area is most advantageous because transport costs (production, supplies, personnel) are minimized, and ventilation airflow routes to the production faces are also minimized.

The number of shafts in a mine depends directly on the daily production rate and the dimensions of the mining area. To obtain a minimum cost per tonne of production, it is essential that an optimum balance between capital expenditure and operating costs be found. In this project, two shafts are considered: a rock hosting shaft (skip shaft) and a ventilation and personnel shaft (service shaft).

Almost all the hard rock mines now have circular shafts because the cross-section provides good geometry for airflow and good rock support characteristics. The circular shutter is easy to move when doing simultaneously lining, resulting in faster work progress during sinking operations /21/.

The lateral cross-section of the shaft is found according to lateral dimensions of the hoisting conveyances and other installations and the designed amount of airflow to fulfil ventilation requirements.

Based on German regulations, the following air quantity requirements and air flow speeds have to be maintained:

- Air quantity requirements
  - Diesel machines: 3.4 m^3/min per kW
  - Humans: 6.0 m^3/min per employee
- Air flow speeds in shafts
  - Air flow speed in the skip shaft: max. 16 m/s
  - Air flow speed in the service shaft: max. 8 m/s

The amount of fresh air required for Golgohar 6 is 13,000 m^3/min. Therefore, the air flow speed in the service shaft will be 7.6 m/sec.

The skip payload capacity will be 2x12 tonnes and the cage could accommodate 20 people. The mine hoist is designed to operate at 15.0 m/s for ore hoisting and 8 m/s for men hoisting. It has a hoisting capacity of 500 t/h of ore.

Hoisting cycle time of skip (Figure 32) is calculated by a winch and skip producer (Deilmann-Haniel), and the general specification of two shafts are listed in Table 33.

CALCULATION OF HOISTING CAPACITY

FOR A DOUBLE-COMPARTMENT SKIP HOISTING INSTALLATION

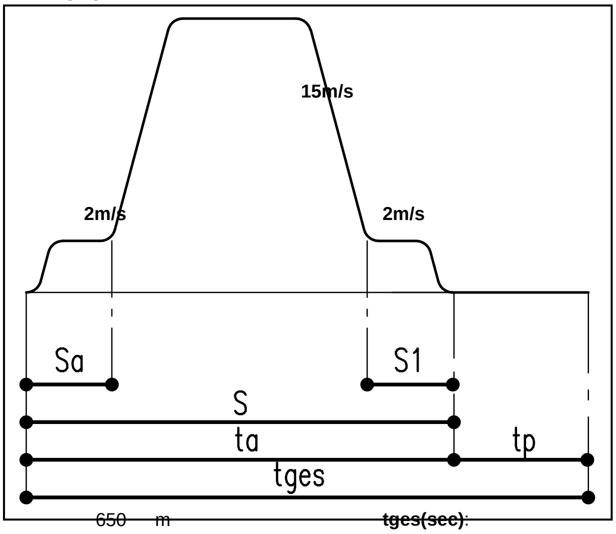
CUSTOMER: ~~Golgohar6 Project~~

PLANT: Sublevel Stoping Shaft

HOISTING CHAR-ACTERISTICS:

HOISTING SPEED	15	m/s	
ACCELERATION:	2	m/s²	WITHOUT CREEPING AT INTERIM LEVELS
DECELERATION	2	m/s²	
JERK LIMITATION	0,9	m/s³	

HOISTING CYCLE



HOISTING DISTANCE	**S:**	650	m	**tges(sec):**	91,67	s
PAUSE:	**tp:**	30	s			
TRAVELLING TIME	**ta:**	61,67	s			
CREEPING:	**Sa:**	3,00	m			
	S1:	15,00	m			

CYCLE TIME: **91,67** **s**

1,5 **Minuten**

SHAFT EFFICIENCY:	**90**	**%**				**MENRIDING**
AVERAGE PAYLOAD:	**12,00**	**t**				**Pers.**
HOURLY CAPACITY:	**471,2**	**t/h**	**>**		**t/h calculated**	**Pers./h**
HOISTING TIME PER DAY:	**20**	**h/d**				
CAPACITY PER DAY:	**8482**	**t/d**	**>**		**t/d nominal**	
OP. DAYS PER YEAR:	**300**	**d/a**				
CAPACITY PER YEAR:	**2.544.669**	**t/a**	**>**		**t/a**	

Figure 32 Calculation of skip hoisting cycle time

Table 33 General data of the shaft systems at Golgohar 6

	Skip Shaft	Service Shaft
Shaft diameter	6.0 m	6.0 m
Shaft depth	650 m	650 m
Production capacity	2x250 t/h	
Hoisting speed	15 m/s	8 m/s
Shaft capacity	10,000 t/d	
Payload	2x12 t	3 t
Motor output	2500 kW	500 kW
Hoisting cycle	92 s	260 s

6.6.3 Transport Drift, Loading Crosscut, Drilling Drift

The development of sublevel stoping in Golgohar 6 will consist of the skip shaft, service shaft, main and service drift, transport drift, loading crosscuts, sublevel drifts, ring drilling drifts and ramp.

Drift excavation is a routine activity in the development of a mine. In mechanized mines, two-boom, electro-hydraulic jumbo drills are used for face drilling. The holes are charged pneumatically with an explosive, usually bulk ammonium nitrate fuel oil (ANFO), from a special charging truck. Short-delay non-electric (NONEL®) detonators are used. Ramp excavation is a routine in the mine development schedule and uses the same equipment as drifting.

Sublevels will be driven in 20 m vertical intervals and will be tied to the footwall ramp system. The sublevels are designed primarily to give access for ring drilling in the stopes.

ANFO, water-gel explosives, emulsions, and heavy ANFOS in bulk or packaged form can all be used for blasting sublevel stoping holes. The selection of an explosive type is largely related to economics. Presently, ANFO is the least expensive form of explosives. ANFO can be free-poured in downholes or pneumatically loaded in upholes.

Ore can be drawn directly into the finger raises or chutes, and then loaded by load-haul-dump units (LHDs) directly out of drawpoints that lie below the cones in cross-cuts.

Mucking is done with LHD vehicles with a bucket capacity of about 6 m^3. Muck is hauled directly to the ore pass system and transferred to the skip shaft loading bin.

Required drifts, crosscuts and ramps in Golgohar 6 are listed in Table 34 .

Table 34 Drifts, Crosscuts and Ramp in Golgohar6

		Length	Size	Section(m^2)
1	**Main Drift**	70 m	7x4	28
2	**Service Drift**	70 m	5x3.5	18
3	**Transport Drift**	600 m	5x3.5	18
4	**Loading Cross-cut**	30x400 m	4x3	12
5	**Sublevel Drift**	4X600 m	4x3	12
6	**Ring Drilling Drift**	30x400 m	3x3	9
7	**Ramp**	1100 m	5x3.5	18

6.7 Interpretation of Results

The following sections provide an interpretation of the results obtained thus far.

6.7.1 Assessment of the Extraction Method Selection

For comparison and evaluation of the three investigated extraction methods (surface mining with shovel and truck, surface mining with crusher and conveyor, sublevel stoping), the calculation of some economical parameters such as internal rate of return (IRR) and net present value (NPV) is needed. First, some assumptions that are common to the three methods are listed in the next section.

6.7.2 Assumptions

- Machinery lifetime is 30,000 hours. Therefore, machinery must be renewed every seven years.
- The first 3 years in all methods are for the opening and development of the mine (without ore production).
- Working time is 20 h/d and 300 d/y.
- Mine lifetime was assumed to be 20 years (after the opening and development period).
- Extraction of iron ore is assumed to be 2,500,000 t/y for all methods.
- The consumption of explosive materials for surface mining and underground mining are 0.350 and 0.650 kg/t respectively.
- Insurance, taxes and governmental customs are considered (18% of interest).
- Auxiliary equipment is considered 10% of equipment prices.
- Transportation and installation of instruments are considered 6% and 5% of equipment prices respectively.

6.7.3 Interpretation of Methods

The machinery prospection and related costs as well as the capital investment for the three methods are estimated in Table 35, Table 36 and Table 37 /23/.

Table 35 Capital cost of sublevel stoping in Golgohar 6

Item	Equipment	Amount	Unit Price(€)	Total Price(€)	Specifications
1	Winch and skip	1	11,000,000	11,000,000	
2	Winch and cage	1	6,000,000	6,000,000	
3	Main ventilator	2	1,700,000	3,400,000	
4	Ventilation instruments		2,500,000	2,500,000	
5	Jumbo drill machine	3	600,000	1,800,000	
6	Ring hole drill machine	4	500,000	2,000,000	
7	Explosive load vehicle	3	200,000	600,000	
8	Scoop tram with rapping arm	1	450,000	450,000	
9	Scoop tram LHD	9	560,000	5,040,000	
10	Roof bolting carriage	1	400,000	400,000	
11	Transport vehicle	4	60,000	240,000	
12	Fuel vehicle	1	200,000	200,000	
13	Dewatering pump and pipes	1	300,000	300,000	
14	Buildings (workshop, store, office)	1,000 m²	1,200	1,200,000	
15	Shaft (6 m diameter)	1,300 m	26,000	33,800,000	Sinking and lining
16	Auxiliary equipment (10%)			3,393,000	For Items 1,2,3,4,5,6,7,8,9,10,11,12,13
17	Transportation (6%)			2,035,800	For Items 1,2,3,4,5,6,7,8,9,10,11,12,13
18	Installation (5%)			1,035,000	For Items 1,2,3,4,13
19	Corrosion protection (0.3%)			69,600	For Items 1,2,3,4,13
20	Contingency (17%)			12,828,778	For the sum of Item 1 to 17
21	Total CAPEX			88,292,178	

Table 36 Capital cost of surface mining by shovel and truck in Golgohar 6

Item	Equipment	Amount	Unit Price(€)	Total Price(€)	Specifications
1	Dump truck	50	920,000	46,000,000	Cat 785C
2	Shovel	5	2,100,000	10,500,000	Liebherr R994
3	DTH drilling machine	2	1,100,000	2,200,000	
4	Explosive load vehicle	3	200,000	600,000	
5	Dewatering pump and pipes	1	200,000	200,000	
6	Fuel vehicle	2	200,000	400,000	
7	Transport vehicle	4	60,000	240,000	
8	Buildings (workshop, store ,office)	1000 m²	1,200	1,200,000	
9	Auxiliary equipment (10%)			6,014,000	For Items 1,2,3,4,5,6,7
10	Transportation (6%)			3,608,400	For Items 1,2,3,4,5,6,7
11	Installation (5%)			10,000	For Item 5
12	Corrosion protection (0.3%)			600	For Item 5
13	Contingency (17%)			10,223,800	For the sum of Items 1 to 12
14	Total CAPEX			81,196,800	

Table 37 Capital cost of surface mining by crushing and conveying in Golgohar 6

Item	Equipment	Amount	Unit Price(€)	Total Price(€)	Specifications
1	Semi-mobile crusher set	2	8,000,000	16,000,000	7,000 t/h
2	Belt conveyor	3,000m	2,100	6,300,000	14,000 t/h,1,800 mm width
3	Dump truck	15	920,000	13,800,000	Cat 785C
4	Shovel	5	2,100,000	10,500,000	Liebherr R994
5	DTH drilling machine	2	1,100,000	2,200,000	
6	Explosive load vehicle	3	200,000	600,000	
7	Dewatering pump and pipe	1	200,000	200,000	
8	Fuel vehicle	1	200,000	200,000	
9	Transport vehicle	4	60,000	240,000	
10	Buildings (workshop, store, office)	1,000m²	1,200	1,200,000	
11	Auxiliary equipment (10%)			5,004,000	For Items 1,2,3,4,5,6,7,8,9
12	Transportation (6%)			2,988,000.84	For Items 1,2,3,4,5,6,7,8,9
13	Installation (5%)			1,125,000	For Items 1,2,7
14	Corrosion protection (0.3%)			67,500	For Items 1,2,7
15	Contingency (17%)			10,272,165.14	For the sum of Items 1 to 14
16	Total CAPEX			70,696,665.98	

Annual operation costs of the sublevel stoping method and both surface mining methods are estimated in Table 35, Table 36 and Table 37. The basis of this estimation and related information and tables (fuel consumption, electric energy consumption, labour cost) can be found in Appendix 5.

Table 38 Operating cost of sublevel stoping in Golgohar 6

Item		Amount	Unit	Cost(EUR/y)	Specifications
1	Consumption of explosives	3,000	t/y	1,950,000	
2	Electric power consumption	44,000,000	kWh/y	2,068,000	
3	Diesel fuel consumption	780,000	l/y	312,000	
4	Lubricant consumption	200	t/y	498,000	
5	Labour costs	176	employees	15,596,000	
6	Service and maintenance costs			7,612,418	10% of machinery capital costs
7	Contingency			5,607,283	20% of Items 1 to 6
8	Total OPEX			33,643,701	

Table 39 Operating cost of surface mining by shovel and truck in Golgohar 6

Item		Amount	Unit	Cost (EUR/y)	Specifications
1	Consumption of explosives	12,000	t/y	7,800,000	
2	Electricity power consumption	1,300,000	kWh/y	61,100	
3	Diesel fuel consumption	13,100,000	l/y	5,240,000	
4	Lubricant consumption	1,900	t/y	4,731,000	
5	Labour costs	276	employees	16,261,000	
6	Service and maintenance costs			4,059,840	5% of machinery capital costs
7	Contingency			5,722,941	15% of Items 1 to 6
8	Total OPEX			43,875,881	

Table 40 Operation Cost of surface Mining by Crushing and Conveying

Item		Amount	Unit	Cost (EUR/y)	Specifications
1	Consumption of explosives	12,000	t/y	7,800,000	
2	Electricity power consumption	56,000,000	kWh/y	2,632,000	
3	Diesel fuel consumption	5,000,000	l/y	2,000,000	
4	Lubricant consumption	1,900	t/y	4,731,000	
5	Labour costs	204	employees	11,851,000	
6	Service and maintenance costs			3,534,833	5% of machinery capital costs
7	Contingency			4,882,325	15% of Items 1 to 6
8	Total OPEX			37,431,158	

Economical parameters are provided by cash flow table preparation. The cash flow table for each method is presented in Appendix 6.

7 Conclusions

The selection of a suitable mining method for Golgohar 6 has been carried out using the qualitative selection method, numerical ranking method and fuzzy set theory and as a result, open pit mining and sublevel stopping have been considered the best choices. A more precise assessment of these two mining methods seems essential in order to make an acceptable comparison between them, which is why both should be evaluated in technical and economic studies. For this special case, the results obtained for the different selection methods are very close, which proves that the correct procedure was used in selecting the mining method.

A comparison of the three methods of extraction is presented in 0, but one of the important items for the economic comparison of projects is product price. IMPASCO has sold some of its extracted iron ore in previous years (the price is shown in Table 41). The next issue is long term pricing. UBS (Union Bank of Switzerland) follows a process to identify long-term inducement prices. They identify all potential projects and their capital and operational costs to determine long-term inducement prices. Furthermore, the prices are expected to revert to their long-term level. In other words, supply and demand will reach equilibrium within 5 years. This is highly dependent on the speed at which the producers bring new capacity to the market.

Based on Mundi index, the price of iron ore fines (62 % Fe spot, CFR Tianjin port) in August 2014 was 100 USD/t (Figure 33), but it is only about half in April 2015..

Table 41 The prices of IMPASCO iron ore(USD/t)

	2009	2010	2011	2012	2013
Iron ore (52% Fe)	16	48	49	42	43
Iron ore (56% Fe)	19	58	60	51	53

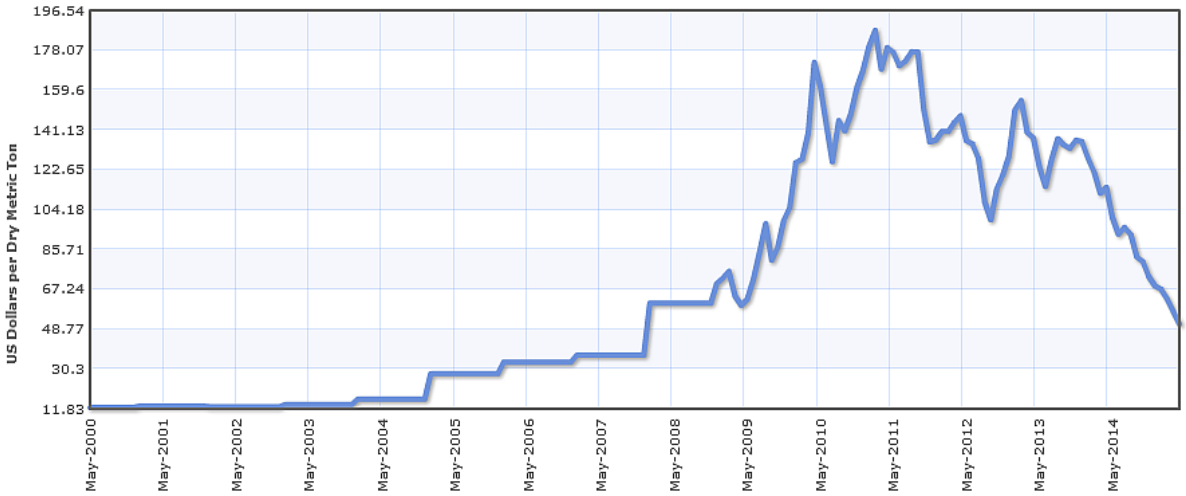


Figure 33 Iron ore price based on Mundi index (62 % Fe fines, CFR Tianjin)

According to IMPASCO's iron ore prices in past years and forecasted worldwide prices, the iron ore price for 56 % Fe will be 40 USD/t for the near future. The economic comparison of the methods is based on this expected price of iron ore.
The result of the cash flow tables and comparison of IRR and NPV(Net Present Value) of all three methods are presented in 0.

Method	CAPEX(€)	CAPEX(€/T)	OPEX(€)	OPEX(€/T)	I.I.R.(%)	N.P.V.(M€)
Surface Mining(Shovel & Truck)	81,196,800	32.5	43,875,881	17.6	18.6	33.4
Surface Mining(Crusher & Conveyor)	70,696,666	28.3	37,431,158	15.0	28.9	91.0
Sublevel Stoping	88,292,178	35.3	33,643,701	13.5	27.4	108.1

Table 42 Comparison of economic parameters

The interest rate of investment financing by banks or financing companies in Iran is 12 percent, therefore with consideration of 12 percent as cost of capital, the Net Present Value of extraction by surface mining (Shovel & truck) will less than the two other methods, and extraction by surface mining (Crusher & conveyor) and sublevel stoping have respectively equal to 91 and 108.1 million Euro.

Although the results of a prefeasibility study typically identify:

- Technical parameters requiring additional examination or test work
- General features and parameters of the given project
- Magnitude of capital and operating cost estimates,

this level of assessment is the preliminary evaluation of a mining project and a conceptual study is useful as a tool to determine if subsequent engineering studies are warranted. However, it is not valid for economic decision making nor is it sufficient for mining method reporting.

The economic analysis of this study is not of sufficient accuracy to access various development options and overall project viability. Therefore, these cost estimates and engineering parameters are typically not considered of sufficient accuracy for final decision making or bank financing. At the prefeasibility study stage, adequate work on geology and mining has been conducted to define mineral resources and mineral reserves of indicated or probable categories, respectively. Sufficient test work and exploration drilling are needed for the final selection of the mining method and for mine developing and processing parameters for equipment selection, flow sheet preparation and production and development scheduling. On the other hand, an experienced team can provide the knowledge base to optimize the project as much as possible or apply the best available proven technology during feasibility study stages.

The level of drilling, sampling and test work must be sufficient to define a proved reserve, flow sheet development, cost estimation and production and engineering design and for more accuracy its necessary to do exploration drilling in a 50x50 m network and taking more samples, geomechanical tests and mineral processing tests in the laboratory and pilot scale.

The evaluation of the Golgohar 6 iron ore deposit with the pattern that is presented in this research shows that the result of the selection for each extraction method is close. This evaluation technique could be applied to other deposits.

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/28/ American Metal Market Events, 2012, World Steel Consumption Assessment, Mexican Steel Forum

/29/ Michael Pascoe, Australian iron ore miners will do well out of shake-out-eventually, September 2014

/30/ IMIDRO, 2014, کلیاتی از وضعیت سنگ آهن و فولاد کشور *معاونت معادن و اکتشافات*

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/32/ Emad Kazemzadeh E., 2014, ساختار بازار خودروي سواري در ايران , Sistan University, Iran

/33/ Pro.Hassan Z. Harraz, Underground Mining, A short series of lectures preparation, Tanta University, 2011

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Appendix 1

- Total intensity map
- Residual magnetic map, reduction to pole
- Residual magnetic map, second vertical derivative
- Residual magnetic map, upward continuation 20m , 40m, 60m
- A1-B1 profile of Golgohar 6

TU Clausthal

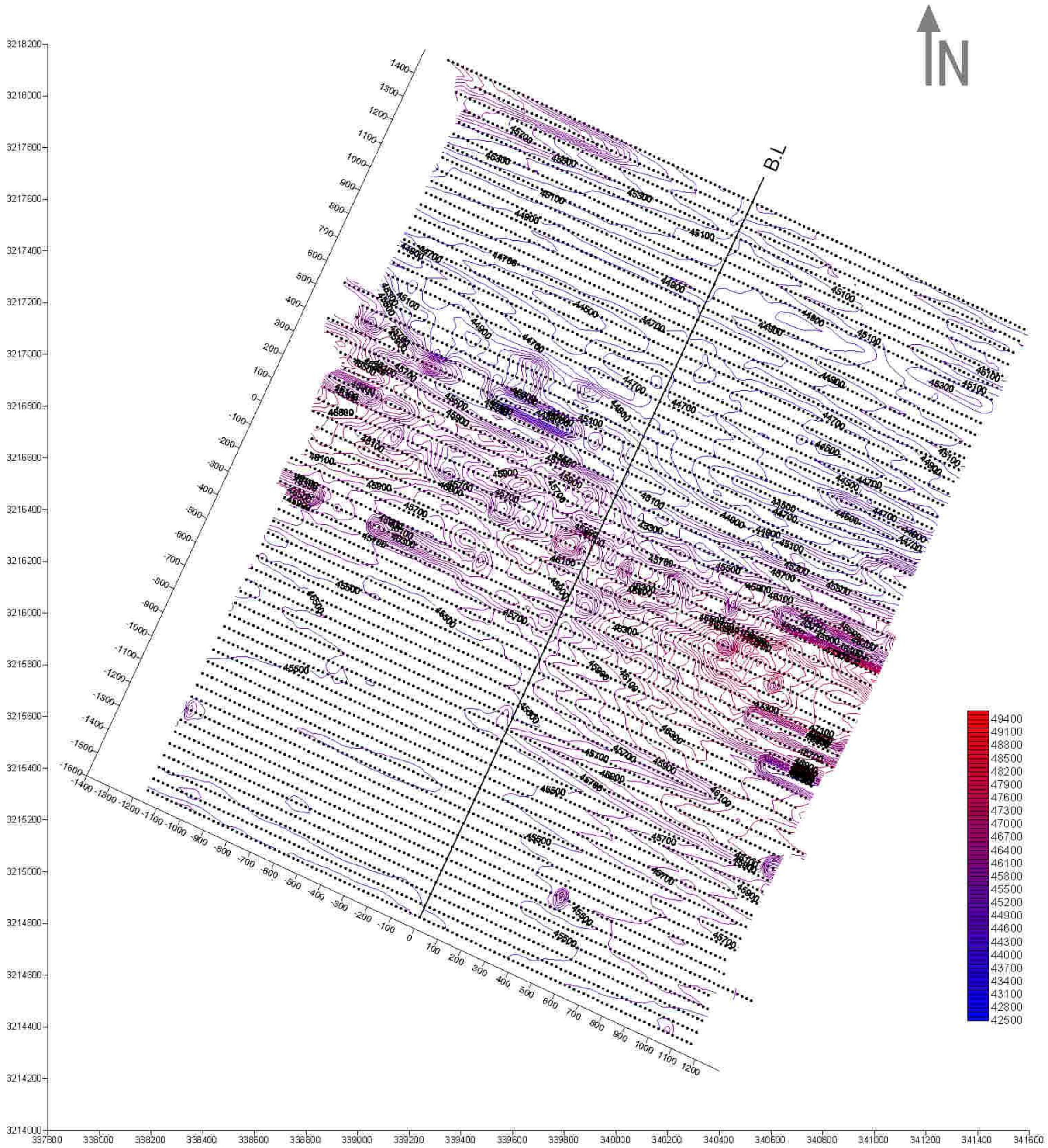
TOTAL INTENSITY MAP (nT)

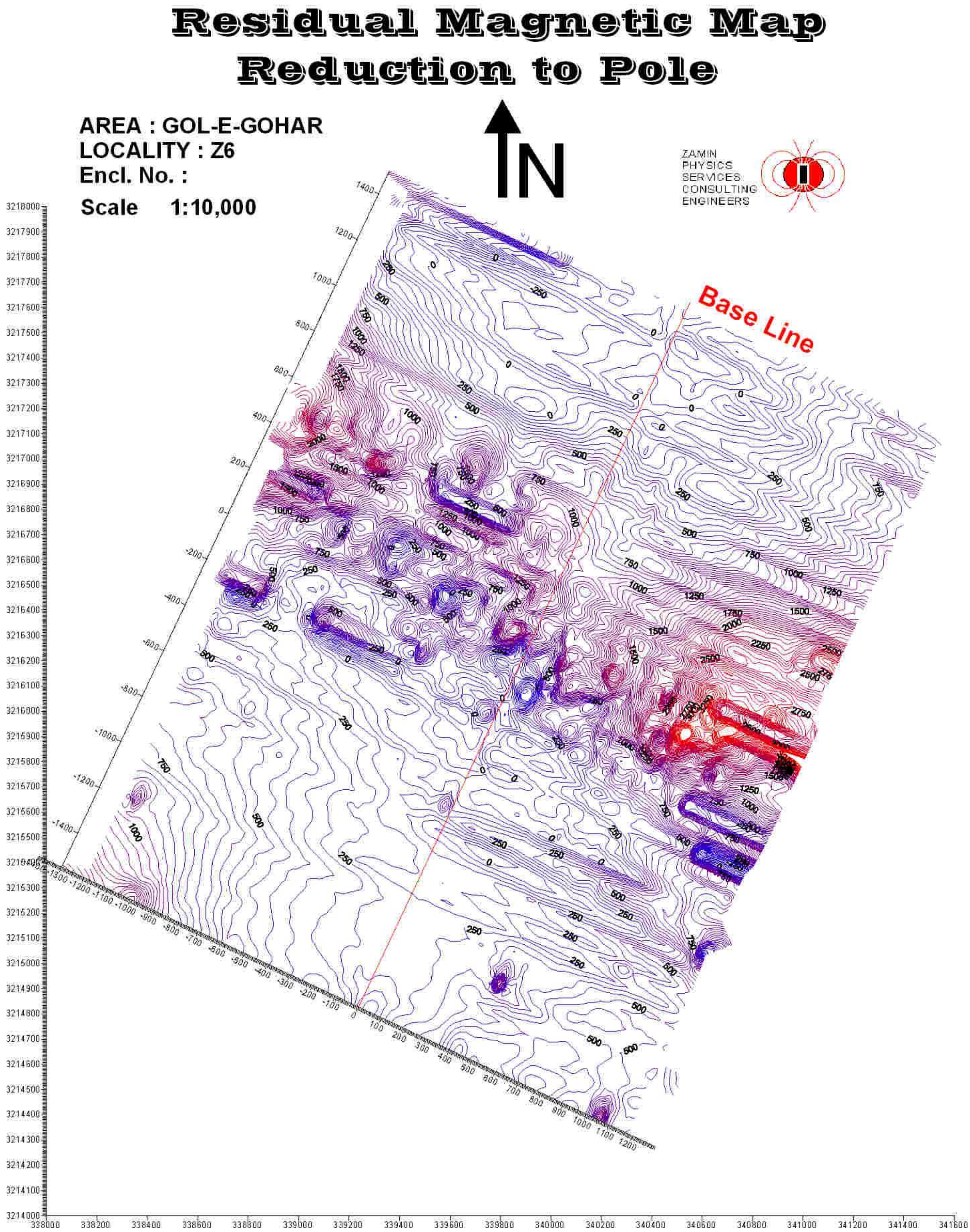
AREA : GOL - E -GOHAR
LOCALITY : Z-6
SCALE = 1:8000

Encl No. :

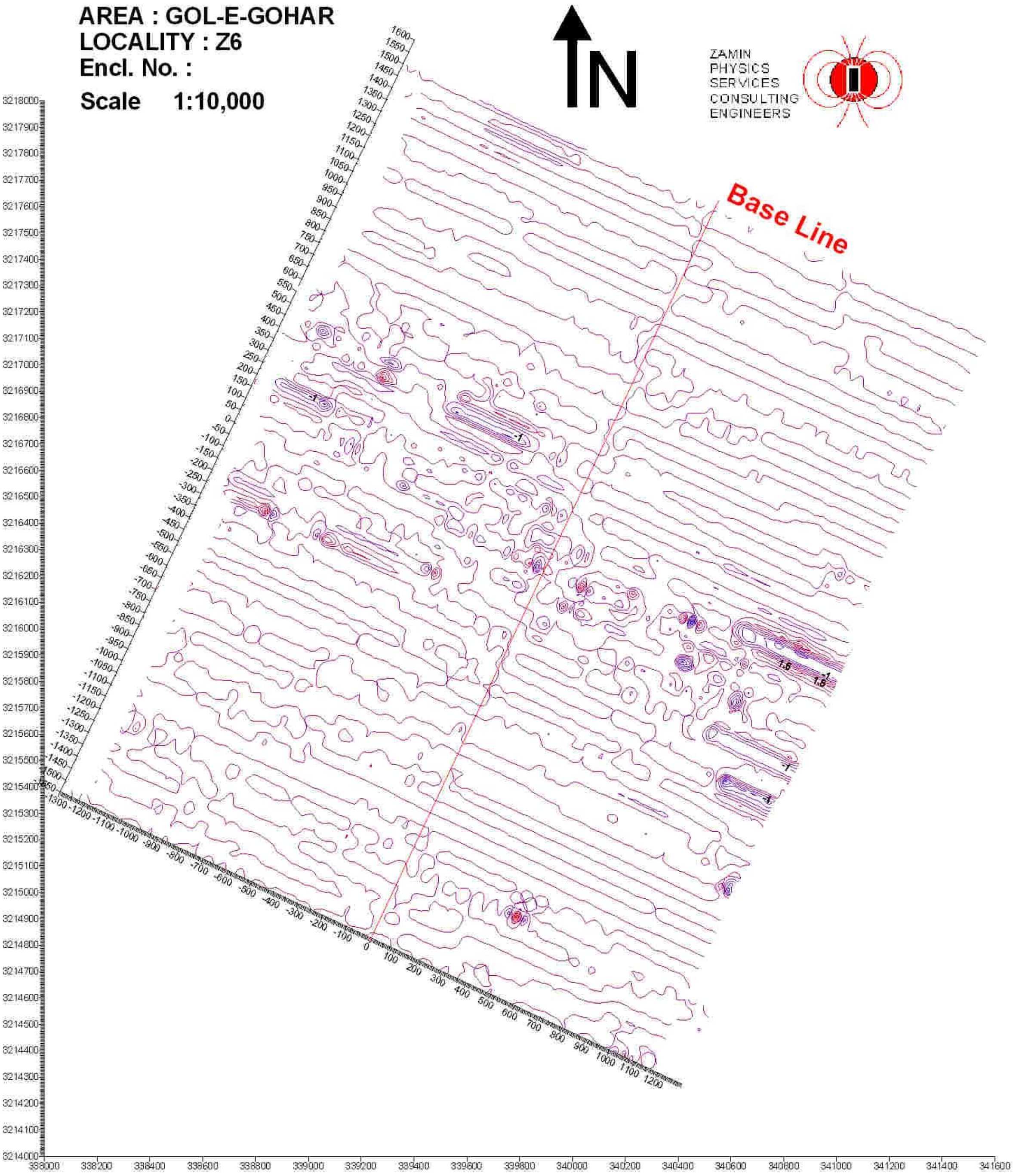
• Points Measured

ZAMIN PHYSICS SERVICES CONSULTING ENGINEERS

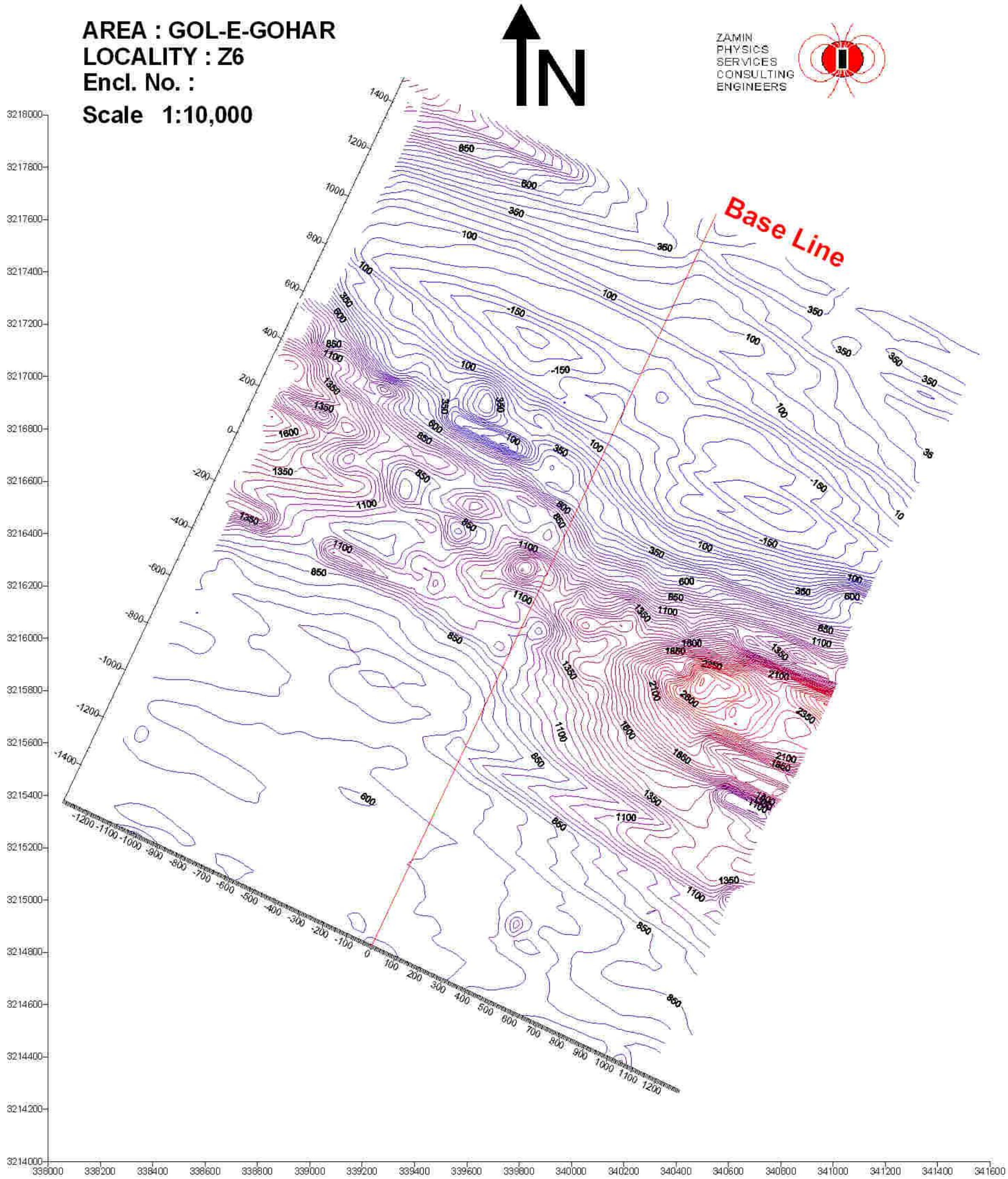


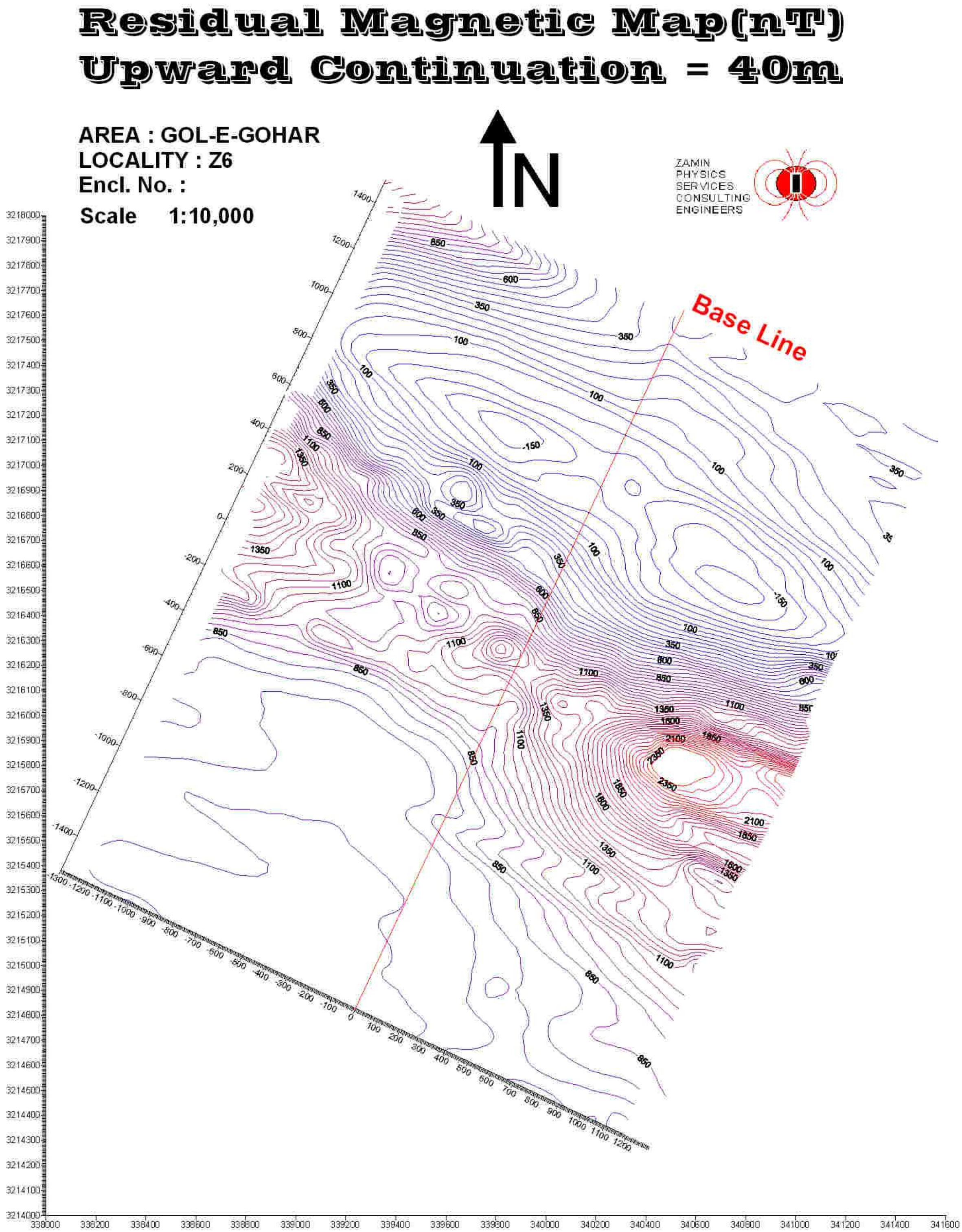

Residual Magnetic Map
Reduction to Pole
AREA : GOL-E-GOHAR
LOCALITY : Z6
Encl. No. :
Scale 1:10,000
N
ZAMIN
PHYSICS
SERVICES
CONSULTING
ENGINEERS
Base Line

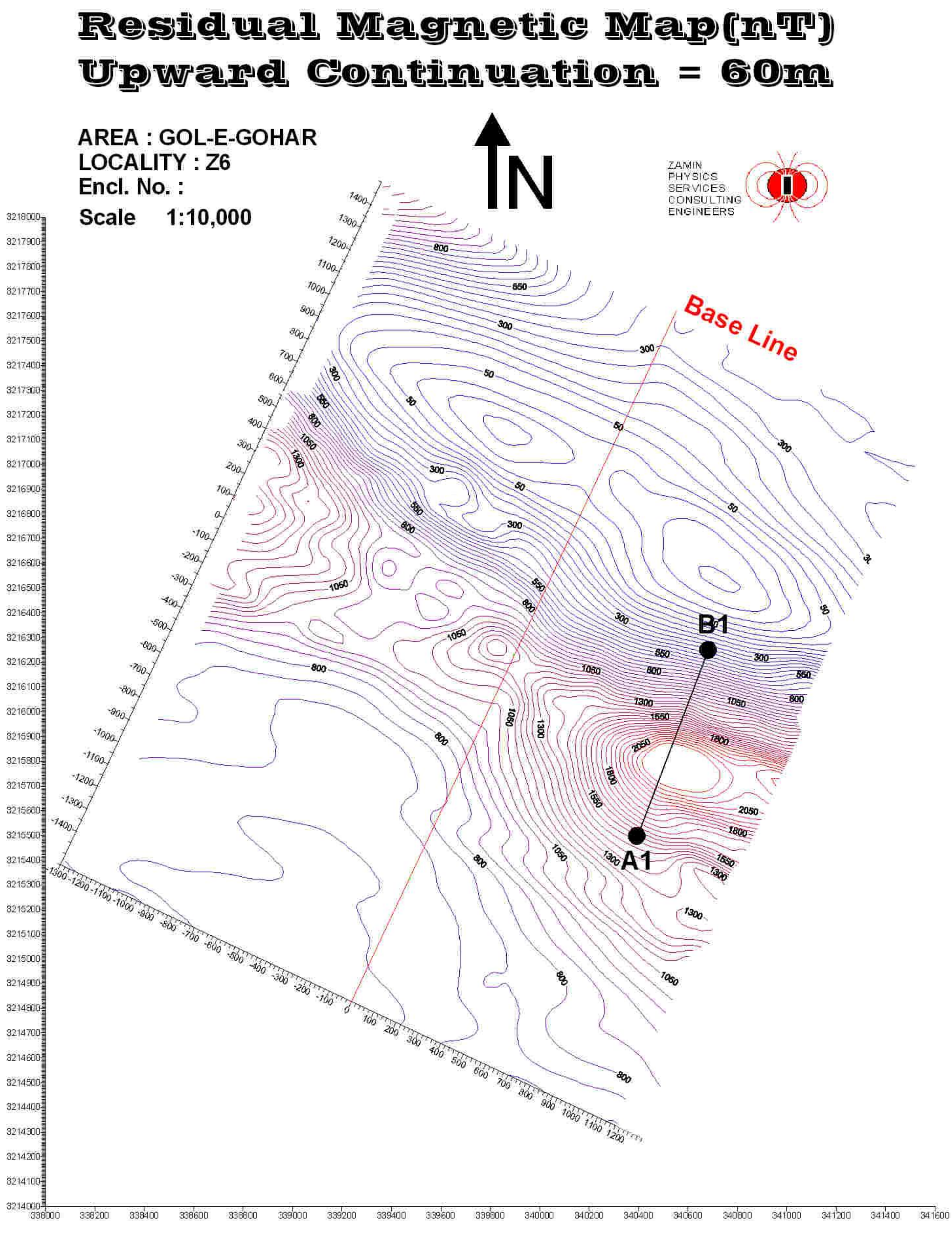
Residual Magnetic Map
Second Vertical Derivative

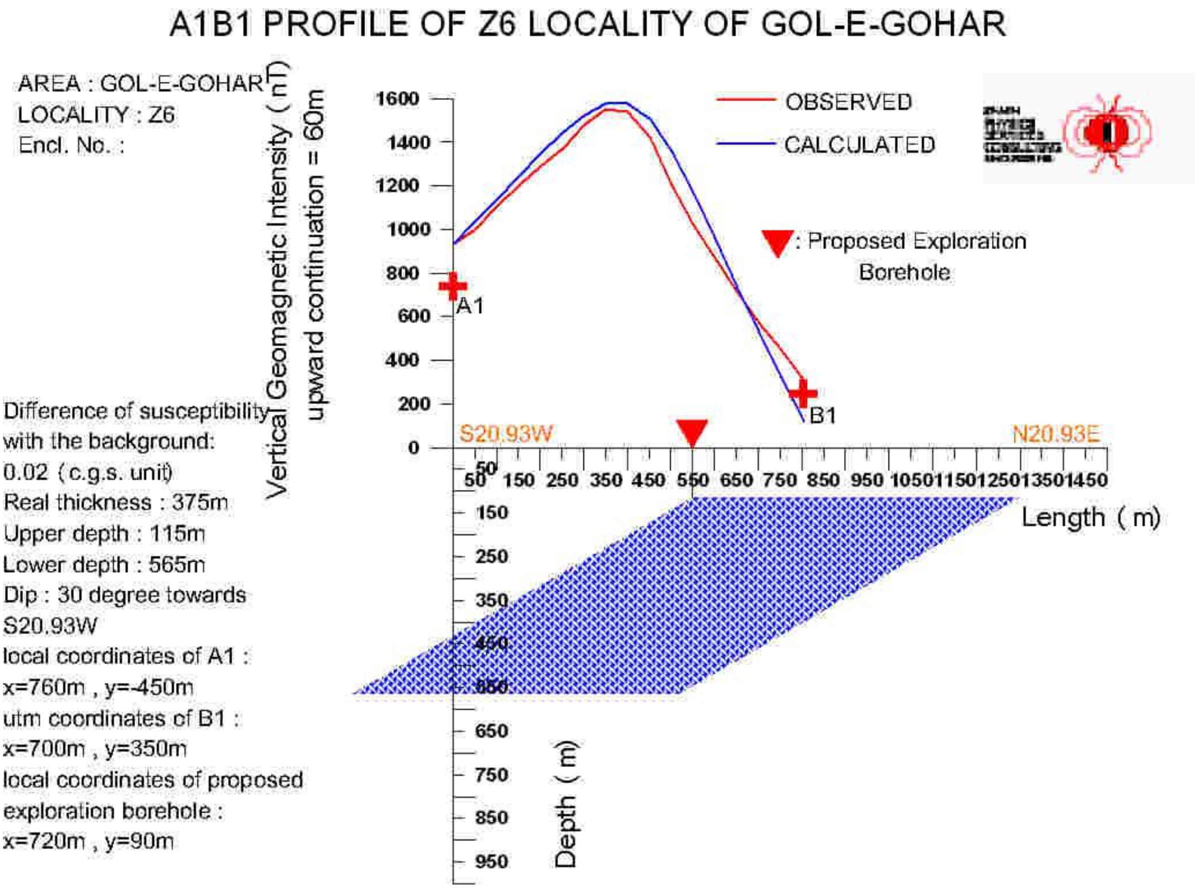


Residual Magnetic Map(nT)
Upward Continuation = 20m




Residual Magnetic Map(nT)
Upward Continuation = 40m
AREA : GOL-E-GOHAR
LOCALITY : Z6
Encl. No. :
Scale 1:10,000
N
ZAMIN
PHYSICS
SERVICES
CONSULTING
ENGINEERS
Base Line


Residual Magnetic Map(nT)
Upward Continuation = 60m
AREA : GOL-E-GOHAR
LOCALITY : Z6
Encl. No. :
Scale 1:10,000
N
ZAMIN
PHYSICS
SERVICES
CONSULTING
ENGINEERS
Base Line
B1
A1


A1B1 PROFILE OF Z6 LOCALITY OF GOL-E-GOHAR
AREA : GOL-E-GOHAR
LOCALITY : Z6
Encl. No. :
Vertical Geomagnetic Intensity (nT)
upward continuation = 60m
OBSERVED
CALCULATED
: Proposed Exploration Borehole
A1
B1
S20.93W
N20.93E
Length (m)
Depth (m)
Difference of susceptibility with the background:
0.02 (c.g.s. unit)
Real thickness : 375m
Upper depth : 115m
Lower depth : 565m
Dip : 30 degree towards S20.93W
local coordinates of A1 :
x=760m , y=-450m
utm coordinates of B1 :
x=700m , y=350m
local coordinates of proposed exploration borehole :
x=720m , y=90m

Appendix 2

- Topographical map of Golgohar 6 area

Golgohar 6 Exploration Area and Topographic Contour Lines

Appendix 3

- Core sampling and sample analysing results

ORE RESERVE
GOL - E - GOHAR Iron Ore
Anomaly no.6

IMPASCO

DATE : 23.06.2010

No.	HOLE NO.	6w002	X = 107011,3	Y = 599582,1		Z = 1746,4	
				Feed			
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
1	498,4	504,4	93	58,74	25,15	0,024	0,871	5,39
2	504,4	510,4	94	62,69	26,21	0,004	0,384	3,73
3	510,4	516,4	95	58,51	25,31	0,024	0,764	6,72
4	516,4	522,4	96	59,79	23,67	0,192	2,925	4,42
5	522,4	528,4	97	57,38	14,37	0,204	1,819	5,83
6	528,4	534,4	98	60,45	23,88	0,218	1,705	3,57
7	534,4	540,4	99	59,57	24,79	0,168	0,794	5,45
8	540,4	546,4	100	43,37	19,35	0,112	0,432	18,15
9	546,4	552,4	101	47,67	21,16	0,091	1,082	10,54
10	552,4	558,4	102	57,59	23,85	0,016	0,561	6,9
11	558,4	564,4	103	60,19	23,70	0,183	1,321	2,62
12	564,4	570,4	104	31,46	13,28	0,126	0,915	11
13	570,4	576,4	105	52,61	19,93	0,088	2,463	1,13
14	576,4	582,4	106	60,23	23,15	0,230	2,285	2,12
15	582,4	588,4	107	64,22	24,93	0,265	0,253	1,31
16	588,4	594,4	108	62,41	25,13	0,306	2,004	1,84
17	594,4	600,4	109	58,23	23,88	0,105	3,430	2,61
18	600,4	606,4	110	63,97	26,39	0,151	2,648	2,2
19	606,4	612,4	111	59,37	23,71	0,173	3,262	3,46
20	612,4	618,4	112	60,76	23,88	0,064	1,445	3,71
21	618,4	624,4	113	56,61	22,99	0,077	1,270	4,97

No.	HOLE NO.	6w003	X = 106936,5	Y = 599515,7		Z = 1747,4	
				Feed			
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
22	477,8	481,4	89	58,13	23,84	0,111	1,611	5,4
23	481,4	487,4	90	61,99	24,77	0,206	1,883	2,5
24	487,4	493,4	91	62,95	26,53	0,141	0,535	3,6
25	493,4	499,4	92	59,65	24,04	0,201	1,983	3,7
26	499,4	505,4	93	60,62	24,37	0,123	2,364	4,9
27	505,4	511,4	94	59,68	24,35	0,233	2,386	3,3
28	511,4	517,4	95	62,58	25,31	0,243	2,792	1,7
29	517,4	523,4	96	61,77	23,48	0,177	2,308	2,4
30	523,4	529,4	97	49,6	20,12	0,187	1,944	9,5
31	529,4	535,4	98	40,7	17,96	0,261	0,990	21,0
32	535,4	541,4	99	54,84	22,98	0,191	1,494	8,9
33	541,4	547,4	100	42,9	17,95	0,039	0,435	17,1
34	547,4	553,4	101	55,8	23,70	0,085	0,301	7,1
35	553,4	559,4	102	60,2	25,65	0,006	0,433	4,6
36	559,4	565,4	103	66,1	27,47	0,001	0,242	2,8
37	565,4	571,4	104	60,1	24,39	0,002	0,279	5,0
38	571,4	577,4	105	66,2	27,11	0,017	0,274	3,6
39	577,4	583,4	106	58,8	24,73	0,161	0,397	7,3
40	583,4	589,4	107	64,4	27,29	0,002	0,258	3,4
41	589,4	595,4	108	63,3	26,20	0,001	0,225	4,75

DATE : 23.06.2010

	HOLE NO.	6w004	X = 107061,000	Y = 599526	Z = 1767,3

No.	FROM	TO	Block no.	Feed % Fe	% Feo	% P	% S	%Sio2
42	526,5	531,3	94	36,1	16,7	0,119	0,399	25,3
43	531,3	537,3	95	62,4	26,7	0,001	0,428	3,2
44	537,3	543,3	96	57,3	25,1	0,022	0,352	7,9
45	543,3	549,3	97	62,3	24,1	0,109	2,174	2,8
46	549,3	555,3	98	58,8	22,8	0,176	3,672	3,1
47	555,3	561,3	99	47,6	19,2	1,010	3,352	9,4
48	561,3	567,3	100	62,8	26,0	0,117	2,399	2,0
49	567,3	573,3	101	57,8	22,6	0,322	1,707	4,2
50	573,3	579,3	102	60,6	23,3	0,153	1,377	4,3
51	579,3	585,3	103	56,3	22,9	0,110	1,491	6,4
52	585,3	591,3	104	26,2	10,8	0,739	3,420	26,5
53	591,3	597,3	105	27,4	12,2	0,463	1,845	25,9
54	597,3	603,3	106	56,3	23,2	0,156	1,304	7,3
55	603,3	609,3	107	57,9	24,4	0,545	1,479	5,8
56	609,3	615,3	108	59,0	25,0	0,035	0,298	6,8
57	615,3	621,3	109	63,4	26,6	0,009	0,374	4,4
58	621,3	627,3	110	61,6	26,4	0,030	0,377	5,1
59	627,3	633,3	111	63,0	26,6	0,022	0,289	5,1
60	633,3	639,3	112	59,0	25,0	0,009	0,336	4,0
61	639,3	641,3	113	64,9	27,8	0,001	0,355	3,8

No.	**HOLE NO.**	**6w005**	**X =**	106728,9	**Y =**	599598,8	**Z =**	1755,8
	FROM	TO	Block no.	Feed % Fe	% Feo	% P	% S	%Sio2
62	489,8	495,8	90	56,9	25,1	0,012	0,459	7,7
63	495,8	501,8	91	63,1	24,8	0,147	2,920	3,0
64	501,8	507,8	92	58,2	22,1	0,136	2,066	5,6
65	507,8	513,8	93	55,8	22,5	0,079	1,220	6,8

No.	**HOLE NO.**	**6w006**	**X =**	106944,88	**Y =**	599656,88	**Z =**	1748
	FROM	TO	Block no.	Feed % Fe	% Feo	% P	% S	%Sio2
66	521,6	524	96	34,5	16,9	0,043	0,495	28,8
67	524	530	97	54,5	22,4	0,226	2,318	8,5
68	530	536	98	55,2	21,5	0,160	1,690	5,5
69	536	539,5	99	54,9	20,8	0,114	3,349	6,7
70	560	566	103	54,8	23,5	0,131	0,221	7,7
71	566	572	104	61,8	24,8	0,079	2,834	2,9
72	572	578	105	51,1	18,3	0,150	0,837	1,5
73	578	584	106	28,8	8,1	0,082	1,301	1,0
74	584	590	107	54,4	17,4	0,016	2,052	1,4
75	590	596	108	59,4	22,8	0,139	0,645	2,0
76	596	602	109	63,4	24,9	0,285	1,624	1,0
77	602	608	110	64,8	25,0	0,280	0,283	1,7

DATE : 23.06.2010

No.	**HOLE NO.**	**6w006**	**X =**	106944,88	**Y =**	599656,88	**Z =**	1748
	FROM	TO	Block no.	Feed % Fe	% Feo	% P	% S	%Sio2
78	608	614	111	57,4	23,7	0,361	0,682	4,1
79	614	620	112	61,7	25,7	0,183	1,983	3,8
80	620	626	113	16,2	9,7	0,086	0,465	44,5
81	626	632	114	55,9	23,3	0,296	0,492	9,2
82	632	638	115	53,9	21,4	0,032	0,603	10,1
83	638	644	116	59,2	25,0	0,014	0,690	6,0

84	644	650	117	55,4	23,0	0,061	0,719	9,3
85	650	656	118	57,4	23,5	0,131	1,364	4,8
86	656	662	119	61,0	24,0	0,322	3,548	3,2
87	662	668	120	55,3	22,4	0,244	3,168	5,7
88	668	671,5	121	54,2	22,8	0,264	2,983	7,4

No.	**HOLE NO.**	**6w007**	**X =**	106828,72	**Y =**	599647,62	**Z =**	1770,93
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
89	507	510,9	90	53,8	23,5	0,091	0,261	9,6
90	510,9	516,9	91	58,1	25,5	0,047	0,534	7,0
91	528,9	534,9	94	42,4	17,6	0,334	2,342	11,9
92	534,9	540,9	95	50,4	21,6	0,530	0,293	10,9
93	540,9	546,9	96	60,0	24,7	0,178	0,245	3,6
94	546,9	552,9	97	58,0	21,2	0,082	0,962	1,6
95	552,9	558,9	98	64,9	25,5	0,412	0,237	1,3
96	558,9	564,9	99	64,3	26,0	0,369	0,317	2,2
97	564,9	570,9	100	62,0	24,1	0,415	0,602	1,8
98	570,9	576,9	101	61,4	24,6	0,283	0,588	2,5
99	576,9	582,9	102	64,5	25,0	0,391	0,471	1,2
100	582,9	588,9	103	64,0	25,1	0,338	0,828	1,0
101	588,9	594,9	104	63,8	25,3	0,378	0,347	1,4
102	594,9	600,9	105	64,7	24,8	0,296	1,333	1,4
103	600,9	606,9	106	63,2	24,4	0,171	1,589	1,9
104	606,9	612,9	107	54,5	20,4	0,313	1,992	5,3
105	612,9	618,9	108	58,9	24,2	0,123	1,904	5,2
106	618,9	624,9	109	59,4	24,8	0,185	2,261	3,8

No.	**HOLE NO.**	**6w008**	**X =**	106861,69	**Y =**	599449,4	**Z =**	1777,41
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
107	469,4	475,4	83	56,3	24,0	0,034	0,534	3,8
108	475,4	481,4	84	50,0	22,3	0,051	0,361	9,5
109	481,4	487,4	85	44,9	21,0	0,034	0,386	21,1
110	487,4	493,4	86	62,2	27,1	0,004	0,337	2,9
111	493,4	499,4	87	59,3	24,8	0,001	0,430	5,3

DATE : 23.06.2010

No.	**HOLE NO.**	**6w009**	**X =**	106870,1	**Y =**	599590,5	**Z =**	1751,07
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
112	486,55	491,7	90	60,9	23,3	0,120	2,865	2,6
113	491,7	497,7	91	47,6	20,6	0,108	1,188	10,4
114	497,7	503,7	92	59,6	25,3	0,143	0,622	5,2
115	503,7	509,7	93	26,6	12,4	0,155	0,476	28,7
116	509,7	515,7	94	58,9	24,9	0,126	1,037	4,1
117	515,7	521,7	95	62,4	25,5	0,167	0,915	3,2
118	521,7	527,7	96	61,6	25,0	0,185	0,225	4,5
119	527,7	533,7	97	51,1	12,0	0,183	0,787	8,6
120	533,7	539,7	98	35,0	16,2	0,059	1,284	25,0
121	539,7	545,7	99	52,8	22,1	0,099	2,698	11,8
122	545,7	551,7	100	52,6	21,9	0,130	2,829	10,3
123	551,7	557,7	101	41,0	16,3	0,080	2,431	18,2
124	557,7	563,7	102	59,6	25,1	0,182	2,355	5,3
125	563,7	569,7	103	61,3	24,9	0,075	0,783	4,0
126	569,7	575,7	104	56,7	22,8	0,477	1,083	4,9
127	575,7	581,7	105	61,0	25,5	0,116	1,649	4,1
128	581,7	587,7	106	61,3	25,8	0,152	2,82	2,8

129	587,7	593,7	107	59,8	24,7	0,151	2,709	5,4
130	593,7	596,13	108	56,5	24,0	0,018	0,569	6,5

No.	HOLE NO.	6w010	X =	106795,48	Y =	599524,3	Z =	1769,82
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
131	507,8	510	90	59,7	21,7	0,237	3,074	2,4
132	510	516	91	34,6	14,2	0,117	0,941	21,7
133	516	522	92	27,3	12,0	0,293	2,712	24,9
134	522	528	93	54,2	23,3	0,184	1,533	8,5
135	528	534	94	34,9	16,1	0,068	0,690	27,0
136	534	540	95	36,4	15,8	0,026	0,802	24,3
137	540	544,1	96	56,2	23,8	0,128	0,280	7,9

No.	HOLE NO.	6w011	X =	106629,680	Y =	599711,3	Z =	1775,63
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
138	528,4	533,6	93	60,0	25,3	0,034	1,060	4,8
139	533,6	539,6	94	58,6	24,7	0,127	1,907	5,6
140	539,6	545,6	95	60,3	21,2	0,180	0,241	2,8
141	545,6	551,6	96	53,4	21,9	0,113	0,243	7,1
142	551,6	557,6	97	61,5	25,1	0,266	0,221	2,9
143	557,6	563,6	98	63,0	26,7	0,301	1,514	3,0
144	575,6	581,6	101	62,0	25,5	0,165	1,949	3,4
145	581,6	587,6	102	63,3	25,3	0,274	1,702	3,3
146	587,6	593,6	103	64,7	26,9	0,239	1,558	2,6
147	593,6	599,6	104	57,6	22,9	0,357	1,636	7,0

DATE : 23.06.2010

No.	HOLE NO.	6w011	X =	106629,680	Y =	599711,3	Z =	1775,63
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
148	599,6	602,5	105	38,9	17,2	0,171	1,606	17,4
149	614	617,6	107	45,7	18,7	0,259	2,182	6,4
150	617,6	623,6	108	57,3	23,1	0,557	2,129	4,9
151	623,6	629,6	109	60,8	24,2	0,585	1,766	4,3
152	629,6	635,6	110	37,0	18,1	0,176	2,623	19,7
153	635,6	641,6	111	18,7	12,2	0,251	0,272	35,9
154	641,6	647,6	112	56,3	24,8	0,137	0,463	5,4
155	647,6	653,6	113	48,3	22,1	0,198	0,545	8,0
156	653,6	659,6	114	50,9	22,6	0,252	0,389	6,0
157	659,6	665,6	115	38,3	15,6	0,075	0,547	7,8
158	665,6	671,6	116	62,9	24,9	0,148	0,395	2,9
159	671,6	677,6	117	62,8	24,9	0,242	0,294	1,6
160	677,6	680	118	58,2	25,1	0,087	0,450	4,1

No.	HOLE NO.	6w013	X =	106784,0	Y =	599779,54	Z =	1792,96
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
161	527,1	533,1	90	57,8	25,0	0,007	0,400	5,8
162	533,1	539,1	91	63,3	27,1	0,001	0,319	4,2
163	539,1	545,1	92	60,1	25,6	0,052	0,879	4,8
164	571,3	575,1	97	50,1	21,2	0,043	0,234	10,2
165	575,1	581,1	98	54,1	20,7	0,063	0,303	9,5
166	581,1	587,1	99	56,2	22,1	0,123	0,431	8,0
167	587,1	593,1	100	60,9	23,3	0,143	0,970	2,1

168	593,1	599,1	101	62,1	24,8	0,152	0,370	3,1
169	599,1	605,1	102	61,2	24,1	0,374	0,697	1,9
170	605,1	611,1	103	63,3	24,2	0,136	0,581	2,3
171	611,1	617,1	104	63,3	25,9	0,124	0,337	2,5
172	617,1	623,1	105	59,9	25,3	0,216	0,408	3,9
173	623,1	629,1	106	49,8	19,9	0,418	0,288	8,9
174	629,1	635,1	107	61,5	23,5	0,125	0,712	3,5
175	635,1	641,1	108	59,7	24,1	0,113	0,288	4,9
176	641,1	647,1	109	62,2	25,8	0,139	0,320	2,9
177	647,1	653,1	110	50,9	21,4	0,150	0,271	10,5
178	653,1	659,1	111	61,0	23,7	0,122	0,326	3,3
179	659,1	665,1	112	64,0	23,5	0,122	0,362	2,7
180	665,1	671,1	113	61,6	24,2	0,218	0,296	3,1
181	671,1	677,1	114	58,4	22,8	0,445	0,915	4,0
182	677,1	683,1	115	58,4	24,2	0,190	0,736	2,6
183	683,1	689,1	116	49,4	20,8	0,010	0,252	10,2
184	689,1	695,1	117	55,6	23,5	0,072	0,379	6,3
185	695,1	701,1	118	58,6	24,9	0,037	0,347	5,4

DATE : 23.06.2010

No.	**HOLE NO.**	**6w014**	**X =**	**106932,60**	**Y =**	599787,61	**Z =**	1773,44
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
186	547,4	549,4	96	49,6	21,0	0,210	0,544	14,9
187	549,4	555,4	97	48,6	21,0	0,041	0,284	11,6
188	555,4	561,4	98	53,0	21,5	0,128	1,195	9,0
189	561,4	567,4	99	47,9	20,4	0,048	0,484	12,9
190	567,4	570,6	100	44,4	19,0	0,664	1,237	11,9
191	615,4	621,4	108	51,6	22,4	0,392	0,264	8,5
192	621,4	627,4	109	57,6	24,4	0,023	0,261	6,2
193	627,4	633,4	110	43,8	18,8	0,096	0,574	11,1
194	633,4	638	111	59,6	24,6	0,029	0,475	5,0
195	645,4	651,4	113	55,2	22,6	0,131	0,337	3,4
196	651,4	657,4	114	56,4	22,6	0,139	0,441	6,1
197	657,4	663,4	115	55,2	21,2	0,553	0,819	4,7
198	663,4	669,4	116	36,7	15,8	0,166	0,529	16,5
199	669,4	675,4	117	39,1	16,9	0,228	0,230	17,2
200	675,4	681,4	118	59,3	22,8	0,112	0,316	9,5
201	681,4	686	119	44,0	20,1	0,087	0,230	13,3

No.	**HOLE NO.**	**6w015**	**X =**	106784,39	**Y =**	599857,8	**Z =**	1784,67
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
202	545,6	548,7	94	53,3	22,4	0,018	1,596	8,7
203	548,7	554,7	95	44,0	19,2	0,155	1,013	16,7
204	554,7	560,7	96	44,4	26,2	0,203	2,134	16,0
205	560,7	566,7	97	52,4	22,2	0,413	0,643	9,7
206	566,7	572,7	98	56,9	24,2	0,633	1,270	5,0
207	572,7	578,7	99	55,9	23,7	0,330	1,196	7,3
208	578,7	584,7	100	47,3	20,1	0,393	2,060	10,0
209	584,7	587,6	101	47,5	21,0	0,376	1,217	15,2
210	638,7	644,7	110	60,4	23,7	0,042	1,157	2,7
211	644,7	650,7	111	61,4	26,2	0,013	0,634	6,2

No.	**HOLE NO.**	**6w016**	**X =**	106535,1	**Y =**	599716,2	**Z =**	1792,33
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
212	580,9	586,3	99	58,7	21,9	0,081	0,425	5,9

213	586,3	592,3	100	60,1	23,1	0,064	1,668	3,9
214	592,3	598,3	101	62,4	23,7	0,105	2,182	2,2
215	598,3	604	102	62,7	21,2	0,271	0,554	2,6
216	625,6	628,3	106	59,4	24,4	0,141	1,661	3,5
217	628,3	634,3	107	65,0	23,3	0,136	1,894	1,4
218	634,3	640,3	108	64,0	26,2	0,199	1,241	1,9
219	640,3	646,3	109	56,3	23,7	0,264	0,556	6,6
220	652,3	658,3	111	59,5	23,0	0,207	0,220	4,4
221	658,3	664,3	112	59,2	23,0	0,351	0,237	3,7

	DATE :	23.06.2010						
No.	**HOLE NO.**	**6w018**	**X =**	106665,36	**Y =**	599801,57	**Z =**	1793,33
						Feed		
	FROM	TO	Block no.	% Fe	% Feo	% P	% S	%Sio2
222	527,7	533,3	90	40,6	18,7	0,020	0,468	23,9
223	533,3	539,3	91	22,2	6,8	0,064	0,693	16,2
224	539,3	545,3	92	57,2	15,6	0,232	1,019	2,8
225	545,3	551,3	93	63,7	24,1	0,057	0,226	1,6
226	551,3	557,3	94	65,0	24,2	0,131	0,233	2,0
227	557,3	563,3	95	28,5	12,2	0,201	0,513	19,7
228	563,3	569,3	96	54,5	22,4	0,194	0,791	4,8
229	569,3	575,3	97	60,8	22,9	0,095	0,218	3,8
230	575,3	581,3	98	63,7	24,2	0,203	0,308	1,6
231	581,3	587,3	99	56,6	23,5	0,243	0,341	5,2
232	587,3	593,3	100	55,2	22,9	0,221	0,297	5,0
233	593,3	599,3	101	60,8	23,5	0,103	0,211	4,0
234	599,3	605,3	102	63,5	24,5	0,232	0,296	2,0
235	605,3	611,3	103	61,0	24,0	0,349	0,856	1,8
236	611,3	617,3	104	51,6	20,3	0,421	0,357	8,4
237	617,3	623,3	105	60,6	23,5	0,047	0,412	2,0
238	623,3	629,3	106	59,6	22,3	0,315	0,863	3,1
239	629,3	635,3	107	63,0	23,9	0,102	0,224	2,9
240	635,3	641,3	108	65,1	24,8	0,056	0,249	1,3
241	641,3	647,3	109	62,3	24,1	0,209	0,246	2,2
242	647,3	653,3	110	56,8	21,4	0,253	0,343	5,3
243	653,3	659,3	111	60,8	22,9	0,328	0,351	3,1
244	659,3	665,3	112	59,1	21,9	0,134	0,263	3,7
245	665,3	671,3	113	62,5	24,6	0,073	0,239	2,8
246	671,3	677,3	114	43,3	18,1	0,037	0,271	16,2
247	677,3	681,5	115	43,6	18,5	0,129	0,248	14,0
248	683,5	689,3	116	50,9	21,7	0,020	0,345	14,9
249	689,3	695,3	117	53,8	22,9	0,017	0,619	9,8
250	695,3	701,3	118	57,8	22,9	0,112	2,635	7,2
251	701,3	707,3	119	52,1	21,9	0,123	3,056	10,9
252	707,3	713,3	120	62,2	25,8	0,061	1,818	4,3
253	713,3	719,3	121	64,7	27,2	0,012	0,603	3,7
254	719,3	725,3	122	62,5	25,6	0,070	0,556	4,1

Appendix 4

- Matrix G calculation based on importance of parameters

	Open pit	Shrinkage	Cut&fill	Room&pillar	Sublevel stoping	Block caving	Sublevel caving	Long wall
Ore strength	80	50	30	10	80	70	70	5
Hanging wall strength	50	20	40	10	30	30	50	10
Foot wall strength	50	25	25	40	20	15	25	40
Deposit shape	75	20	25	15	55	70	60	20
Deposit dip	60	35	40	30	45	55	35	15
Deposit size	50	30	20	25	60	65	60	30
Ore grade	70	50	35	40	75	75	75	40
Ore uniformity	30	20	20	15	40	25	20	15
Depth	20	30	35	30	65	70	60	40
Ore RMR	50	50	40	55	60	30	55	25
Hanging wall RMR	55	35	60	20	65	70	55	50
Foot wall RMR	30	35	40	30	35	30	25	30
Ore thichness	40	25	15	10	60	50	50	10
Subsidence	70	30	50	40	30	25	60	25
Reccovery	45	50	60	35	50	40	40	50
Selectivity	60	30	40	55	50	35	45	30
Dilution	50	50	70	65	45	30	30	60
Environmental risk	10	40	45	40	45	40	15	30
Production rate	30	25	20	35	55	60	50	40
Flexibility	50	40	35	65	60	30	55	30
Grade distribution	40	40	55	40	45	35	40	20
Investment	20	30	25	40	30	30	20	25
Operating costs	15	40	35	35	45	50	40	30

Max = 80 , Min = 5

In the matrix G Max = 0.504 , Min = 0.0117

	Open pit	Shrinkage	Cut&fill	Room&pillar	Sublevel stoping	Block caving	Sublevel caving	Long wall
Ore strength	0,504	0,3071	0,1758	0,0445	0,504	0,4384	0,4384	0,0117
Hanging wall strength	0,3071	0,1102	0,2414	0,0445	0,1758	0,1758	0,3071	0,04452
Foot wall strength	0,3071	0,143	0,143	0,2414	0,1102	0,0773	0,143	0,24144
Deposit shape	0,4712	0,1102	0,143	0,0773	0,3399	0,4384	0,3727	0,11016
Deposit dip	0,3727	0,2086	0,2414	0,1758	0,2743	0,3399	0,2086	0,07734
Deposit size	0,3071	0,1758	0,1102	0,143	0,3727	0,4055	0,3727	0,1758
Ore grade	0,4384	0,3071	0,2086	0,2414	0,4712	0,4712	0,4712	0,24144
Ore uniformity	0,1758	0,1102	0,1102	0,0773	0,2414	0,143	0,1102	0,07734
Depth	0,1102	0,1758	0,2086	0,1758	0,4055	0,4384	0,3727	0,24144
Ore RMR	0,3071	0,3071	0,2414	0,3399	0,3727	0,1758	0,3399	0,14298
Hanging wall RMR	0,3399	0,2086	0,3727	0,1102	0,4055	0,4384	0,3399	0,30708
Foot wall RMR	0,1758	0,2086	0,2414	0,1758	0,2086	0,1758	0,143	0,1758
Ore thichness	0,2414	0,143	0,0773	0,0445	0,3727	0,3071	0,3071	0,04452
Subsidence	0,4384	0,1758	0,3071	0,2414	0,1758	0,143	0,3727	0,14298
Reccovery	0,2743	0,3071	0,3727	0,2086	0,3071	0,2414	0,2414	0,30708
Selectivity	0,3727	0,1758	0,2414	0,3399	0,3071	0,2086	0,2743	0,1758
Dilution	0,3071	0,3071	0,4384	0,4055	0,2743	0,1758	0,1758	0,37272
Environmental risk	0,0445	0,2414	0,2743	0,2414	0,2743	0,2414	0,0773	0,1758
Production rate	0,1758	0,143	0,1102	0,2086	0,3399	0,3727	0,3071	0,24144
Flexibility	0,3071	0,2414	0,2086	0,4055	0,3727	0,1758	0,3399	0,1758
Grade distribution	0,2414	0,2414	0,3399	0,2414	0,2743	0,2086	0,2414	0,11016
Investment	0,1102	0,1758	0,143	0,2414	0,1758	0,1758	0,1102	0,14298
Operating costs	0,0773	0,2414	0,2086	0,2086	0,2743	0,3071	0,2414	0,1758

Appendix 5

- Operation costs of extraction by Sublevel stoping and Open pit mining

Operation Cost of Sublevet Stoping in Golgohar6

Opex Estimation

Item		Amount	Unit	Cost(Euro/Year)	
1	Consumption of Explosive	3000	Tonnes/Year	1950000	
2	Electricity Power Consumption	44000000	KWh/Year	2068000	
3	Diesel Fuel Consumption	780000	Li/Year	312000	
4	Lubricant Consumption	200	Tonnes/Year	498000	
5	Labour costs	176	Employees	15596000	
6	Service and Maintenance costs			7612417,8	10% of Machinary Capital costs
7	Contingency			5607283,56	20% of Item 1 to 6
8	Total OPEX			33643701,36	

Diesel Fuel Consumption	KW	No.	KWh/Year
Jumbo Drill Machine	90	3	1296000
Ring Hole Drill Machine	85	4	1632000
Explosive Load Vehicle	105	3	1512000
Scoop Term With Rapping Arm	176	1	844800
Scoop Term LHD	170	9	7344000
Roof Bolting Carriage	85	1	408000
Transport Vehicle	65	4	1248000
Fuel Vehicle	120	1	576000
		Total	14860800
		Li/Year	772761,6

Labour costs	No.	Salary(€/Mo)	
Winch and Skip	16	4500	72000
Winch and Cage	8	4500	36000
Main Ventilator	8	4500	36000
Ventilation instruments	12	4500	54000
Jumbo Drill Machine	12	4500	54000
Ring Hole Drill Machine	15	4500	67500
Explosive Load Vehicle	24	4500	108000
Scoop Term With Rapping Arm	4	4500	18000
Scoop Term LHD	27	4500	121500
Roof Bolting Carriage	8	4500	36000
Transport Vehicle	12	4500	54000
Fuel Vehicle	4	4500	18000
Dewatering Pump and Pipe	8	4500	36000
Lighting andElectrical Technici-an	12	4500	54000
Mechanical Technician	12	4500	54000
Store Responsible	8	4000	32000
Guard	6	4000	24000
Other Services(Worker)	20	4000	80000
Upperlevel Manager	3	8000	24000
Middle level manager	9	7000	63000
Engineer	12	6000	72000
Total	240		1114000
		Euro/Year	15596000

Operation Cost of surface Mining by Shovel and Truck in Golgohar6

Item		Amount	Unit	Cost(Euro/Year)	
1	Consumption of Explosive	12000	Tonnes/Year	7800000	
2	Electricity Power Consumption	1300000	KWh/Year	61100	
3	Diesel Fuel Consumption	13100000	Li/Year	5240000	
4	Lubricant Consumption	1900	Tonnes/Year	4731000	
5	Labour costs	276	Employees	16261000	
6	Service and Maintenance costs			4059840	5% of Machinary Capital costs
7	Contingency			5722941	15% of Item 1 to 6
8	Total OPEX			43875881	

Diesel Fuel Consumption	KW	No.	KWh/Year
Dump Truck	934	50	224160000
Shovel	949	5	22776000
DTH Drilling Machine	100	2	960000
Explosive Load Vehicle	105	3	1512000
Transport Vehicle	65	4	1248000
Fuel Vehicle	120	2	1152000
		Total	251808000
		Li/Year	13094016

Labour costs	No.	Salary(€/Mo)	
Dump Truck operator	140	4500	630000
Shovel operator	20	4500	90000
DTH Drilling Machine operator	12	4500	54000
Explosive Load Vehicle operator	18	4500	81000
Dewatering Pump and Pipe responsible	3	4500	13500
Fuel Vehicle operator	6	4500	27000
Transport Vehicle operator	14	4500	63000
Electrical Technician	6	4500	27000
Mechanical Technician	6	4500	27000
Store Responsible	3	4000	12000
Guard	6	4000	24000
Other Services(Worker)	12	4000	48000
Upperlevel Manager	1	8000	8000
Middle level manager	3	7000	21000
Engineer	6	6000	36000
Total	256		1161500
		Euro/Year	16261000

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Operation Cost of surface Mining by Crushing and Conveying in Golgohar6

Item		Amount	Unit	Cost(Euro/Year)	
1	Consumption of Explosive	12000	Tonnes/Year	7800000	
2	Electricity Power Consumption	56000000	KWh/Year	2632000	
3	Diesel Fuel Consumption	5000000	Li/Year	2000000	
4	Lubricant Consumption	1900	Tonnes/Year	4731000	
5	Labour costs	204	Employees	11851000	
6	Service and Maintenance costs			3534833,3	5% of Machinary Capital costs
7	Contingency			4882324,995	15% of Item 1 to 6
8	Total OPEX			37431158,3	

Diesel Fuel Consumption	KW	No.	KWh/Year
Dump Truck	934	15	67248000
Shovel	949	5	22776000
DTH Drilling Machine	100	2	960000
Explosive Load Vehicle	105	3	1512000
Transport Vehicle	65	4	1248000
Fuel Vehicle	120	2	1152000
		Total	94896000
		Li/Year	4934592

Labour costs	No.	Salary(€/Mo)	
Dump Truck operator	40	4500	180000
Shovel operator	20	4500	90000
Semi Mobile Crusher Set	18	4500	81000
Belt Conveyor	15	4500	67500
DTH Drilling Machine operator	12	4500	54000
Explosive Load Vehicle operator	18	4500	81000
Dewatering Pump and Pipe responsible	3	4500	13500
Fuel Vehicle operator	3	4500	13500
Transport Vehicle operator	14	4500	63000
Electrical Technician	6	4500	27000
Mechanical Technician	6	4500	27000
Store Responsible	3	4000	12000
Guard	6	4000	24000
Other Services(Worker)	12	4000	48000
Upperlevel Manager	1	8000	8000
Middle level manager	3	7000	21000
Engineer	6	6000	36000
Total	186		846500
		Euro/Year	11851000

Appendix 6

- Economical parameters calculation

Cash Flow of Sublevel Stoping in Golgohar6

Year	0	1	2	3	4	5	6	7	8	9
Production per year(tonnes)				2500000	2500000	2500000	2500000	2500000	2500000	2500000
Price of products (€/tonnes)				30	30	30	30	30	30	30
Cash in (€)				75000000	75000000	75000000	75000000	75000000	75000000	75000000
Investment (€)	29430726	29430726	29430726							18390294
Operation cost (€)				33643701	33643701	33643701	33643701	33643701	33643701	33643701
				9098386	9098386	9098386	9098386	9098386	9098386	
Cash flow (€)	-29430726	-29430726	-29430726	32257913	32257913	32257913	32257913	32257913	32257913	22966005
NPV (€)	-29430726	-23101041	-18132685	15600122	12244993	9611455	7544313	5921753	4648158	2597530

10	11	12	13	14	15	16	17	18	19	20	21	22
2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000
30	30	30	30	30	30	30	30	30	30	30	30	30
75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000
						18390294						
33643701	33643701	33643701	33643701	33643701	33643701	33643701	33643701	33643701	33643701	33643701	33643701	33643701
9098386	9098386	9098386	9098386	9098386	9098386		9098386	9098386	9098386	9098386	9098386	9098386
32257913	32257913	32257913	32257913	32257913	32257913	22966005	32257913	32257913	32257913	32257913	32257913	32257913
2863795	2247877	1764425	1384949	1087087	853286	476842	525722	412654	323905	254242	199562	156642

IRR=27.4% , NPV=108.1 M€

Cash Flow of Surface Mining with Shovel and Truck in Golgohar6

Year	0	1	2	3	4	5	6	7	8	9
Production per year(tonnes)				2500000	2500000	2500000	2500000	2500000	2500000	2500000
Price of products (€/tonnes)				30	30	30	30	30	30	30
Cash in (€)				75000000	75000000	75000000	75000000	75000000	75000000	75000000
Investment (€)	27065600	27065600	27065600							79996800
Operation cost (€)				43875881	43875881	43875881	43875881	43875881	43875881	43875881
				6847306	6847306	6847306	6847306	6847306	6847306	
Cash flow (€)	-27065600	-27065600	-27065600	24276813	24276813	24276813	24276813	24276813	24276813	-48872681
NPV (€)	-27065600	-22820911	-19241915	14552500	12270236	10345899	8723355	7355274	6201749	-10526984

10	11	12	13	14	15	16	17	18	19	20	21	22
2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000
30	30	30	30	30	30	30	30	30	30	30	30	30
75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000
						79996800						
43875881	43875881	43875881	43875881	43875881	43875881	43875881	43875881	43875881	43875881	43875881	43875881	43875881
6847306	6847306	6847306	6847306	6847306	6847306		6847306	6847306	6847306	6847306	6847306	6847306
24276813	24276813	24276813	24276813	24276813	24276813	-48872681	24276813	24276813	24276813	24276813	24276813	24276813
4409048	3717578	3134551	2642961	2228466	1878976	-3189416	1335832	1126334	949692	800752	675170	569283

IRR=18.6% , NPV=33.4 M€

Cash Flow of Surface Mining with Crushing and Conveying in Golgohar6

Year	0	1	2	3	4	5	6	7	8	9
Production per year(tonnes)				2500000	2500000	2500000	2500000	2500000	2500000	2500000
Price of products (€/tonnes)				30	30	30	30	30	30	30
Cash in (€)				75000000	75000000	75000000	75000000	75000000	75000000	75000000
Investment (€)	23565555	23565555	23565555							46585305
Operation cost (€)				37431158	37431158	37431158	37431158	37431158	37431158	37431158
				8265145	8265145	8265145	8265145	8265145	8265145	
Cash flow (€)	-23565555	-23565555	-23565555	29303697	29303697	29303697	29303697	29303697	29303697	-9016463
NPV (€)	-23565555	-18282044	-14183122	13682443	10614774	8234890	6388588	4956236	3845024	-917825

10	11	12	13	14	15	16	17	18	19	20	21	22
2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000	2500000
30	30	30	30	30	30	30	30	30	30	30	30	30
75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000	75000000
						46585305						
37431158	37431158	37431158	37431158	37431158	37431158	37431158	37431158	37431158	37431158	37431158	37431158	37431158
8265145	8265145	8265145	8265145	8265145	8265145		8265145	8265145	8265145	8265145	8265145	8265145
29303697	29303697	29303697	29303697	29303697	29303697	-9016463	29303697	29303697	29303697	29303697	29303697	29303697
2314159	1795314	1392796	1080524	838265	650322	-155235	391402	303648	235568	182753	141779	109991

IRR=28.9% , NPV=91.0 M€